





THE UNIVERSITY  
OF ILLINOIS  
LIBRARY

622.09

Un 32 b

No. 105-112

BLDG. HALL



The person charging this material is responsible for its return on or before the **Latest Date** stamped below.

Theft, mutilation and underlining of books are reasons for disciplinary action and may result in dismissal from the University.

UNIVERSITY OF ILLINOIS LIBRARY AT URBANA-CHAMPAIGN

BUILDING USE ONLY

L161 5 T WDC

MAR 27 1995

MAR 27 1995

JUL 24 1996

2-16-97

FEB 20 1997

BUILDING USE ONLY

SEP 09 1999

L161—O-1096

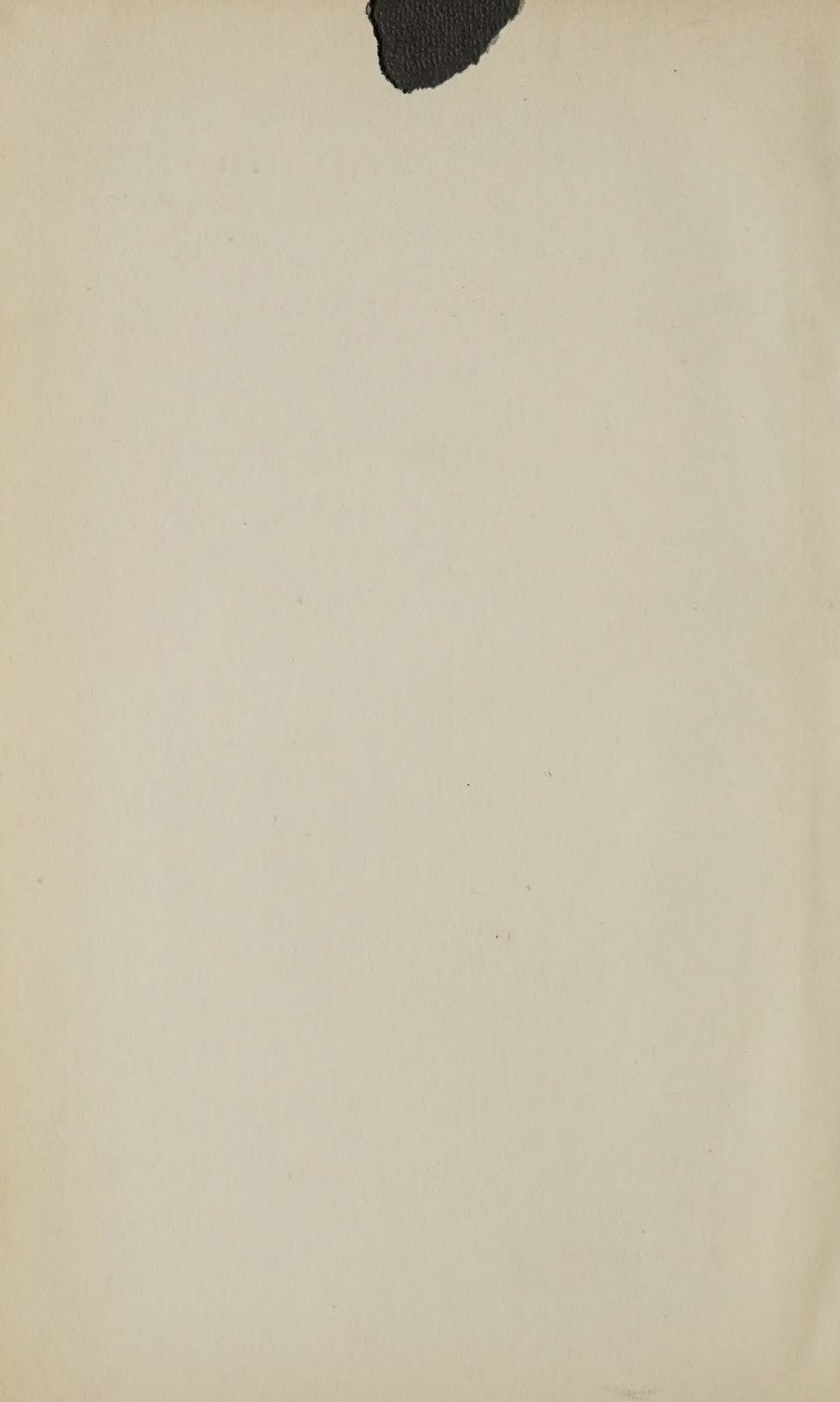














DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

BLACK DAMP IN MINES

BY

G. A. BURRELL, I. W. ROBERTSON

AND

G. G. OBERFELL



WASHINGTON  
GOVERNMENT PRINTING OFFICE  
1916 *h.c.*

The Bureau of Mines, in carrying out one of the provisions of its organic act—to disseminate information concerning investigations made—prints a limited free edition of each of its publications.

When this edition is exhausted, copies may be obtained at cost price only through the Superintendent of Documents, Government Printing Office, Washington, D. C.

The Superintendent of Documents *is not an official of the Bureau of Mines*. His is an entirely separate office and he should be addressed:

SUPERINTENDENT OF DOCUMENTS,  
*Government Printing Office,  
Washington, D. C.*

The general law under which publications are distributed prohibits the giving of more than one copy of a publication to one person. The cost of this publication is 10 cents.

*First edition. June, 1916.*



622.09  
Un32b  
no. 105-112

*Engin. lib.*

## CONTENTS.

	Page.
Introduction.....	5
Comparison of atmospheric air with mine air.....	5
Composition of the atmosphere.....	6
Transfer of oxygen and carbon dioxide in breathing.....	7
Effects on man of variations in composition and amount of air.....	7
Effect of carbon dioxide.....	7
Effect of diminished oxygen supply.....	9
Effect of air at high altitudes.....	10
Effect of breathing a decreasing oxygen supply.....	11
Effect of a mine atmosphere low in oxygen.....	11
Effect of atmospheres low in oxygen on canaries and mice.....	12
General comments on tests.....	14
Effects of oxygen deficiency as showing relative value of birds and mice in exploring mines.....	14
Effect of temperature and moisture on man.....	15
Ventilation of buildings above ground.....	15
Ventilation conditions in mines.....	16
Action of oxygen on coal.....	19
Influence of moisture on rate of absorption of oxygen by coal.....	20
Absorption of oxygen and production of carbon dioxide from atmospheric air by wood.....	21
Discussion of results.....	22
Effect of lowering oxygen and raising carbon dioxide content on explosibility of methane-air mixtures.....	23
Consumption of oxygen and production of carbon dioxide by men and by lights.....	25
Fouling of mine air by decay of timber.....	26
Specific gravity and composition of black damp.....	26
Effect on lights of air low in oxygen and high in carbon dioxide.....	28
Effect of atmospheres low in oxygen on fires in mines.....	31
Results of analyses of coal-mine atmospheres.....	32
Results of analyses of samples in series 1.....	33
Comment on analyses of samples in series 1.....	42
Temperature and humidity.....	43
Results of analyses of samples in series 2 to 6.....	43
Comment on results of sampling and analysis.....	48
Samples in series 2.....	48
Samples in series 3.....	48
Samples in series 4.....	48
Samples in series 5.....	48
Samples in series 6.....	49
Effects of temperature and humidity and other factors.....	49
Effects of leakage of air through doors, brattices, etc.....	50
Inflammable gas in mines.....	50

	Page.
Occurrence and composition of gas in rock strata of the Cripple Creek gold-mining district, Colorado.....	51
Origin of the strata gas.....	51
Effects of barometric pressure on outflow of strata gas.....	52
Collection of samples of gas in four metal mines.....	52
Observations on gas in Midget mine.....	52
General observations regarding gaseous conditions in Anaconda mine..	53
Observations on gas in Cresson mine.....	53
Tabulated data.....	54
Special air samples from Mary McKinney mine.....	58
Composition of strata gas samples calculated on air-free basis.....	60
Effect on men of partial pressures of oxygen.....	61
Comparison between the indications afforded by candle and by acetylene flame and analyses of the samples.....	62
Combustible gas in the rock-strata gas.....	62
Ventilation conditions of 15 metal mines as indicated by analyses of air samples.....	63
Comments on results of analysis.....	76
Observations regarding black damp in certain metal mines.....	77
Summary.....	78
Publications on mine accidents and methods of coal mining.....	82
Index.....	85



# BLACK DAMP IN MINES.

---

By G. A. BURRELL, I. W. ROBERTSON, AND G. G. OBERFELL.

---

## INTRODUCTION.

The Bureau of Mines, in pursuing investigations looking to greater safety in mining, has analyzed samples of the air in many different coal mines in the United States, and has studied the analyses. This report presents the results of one phase of this study and shows how atmospheric air, after entering a coal mine, loses oxygen and gains carbon dioxide with resulting formation of so-called black damp. Also the report discusses the effects of the constituents of black damp on men, on the burning of oil and acetylene lamps, and on the explosibility of methane.

The term "black damp" was and still is widely used to designate accumulations of carbon dioxide, but a more exact definition of black damp, as Haldane, the English physiologist, has pointed out, is an accumulation of carbon dioxide and nitrogen in proportions larger than those found in atmospheric air. The reasons for preferring Haldane's definition are given on succeeding pages.

## COMPARISON OF ATMOSPHERIC AIR WITH MINE AIR.

When atmospheric air enters a coal mine, it changes in composition according to (1) the velocity with which it traverses the workings; (2) the amount of coal with which it comes in contact—that is, the extent of the mine workings that it traverses; (3) the gaseous nature of the seam; (4) the tendency of the coal to absorb oxygen; and (5) the temperature and wetness of the mine.

As regards the details mentioned, the governing conditions are as follows: (1) Other things being equal, more carbon dioxide and methane are present and there is a greater deficiency of oxygen at places where the air is still, as at working faces and in old workings. (2) The purity of the air depends on the distance it has traveled, so that in a well-ventilated mine the air is purer at working faces near the shaft than at those that are remote. In the same mine the air will be fresher and purer when the mine is ventilated by a split system than when the air traverses each working face and entry. (3) Some mines vary greatly as regards the generation of methane. Methane is not only

dangerously inflammable, but, if introduced in large proportions at some parts of a mine, it lowers the oxygen content of the atmosphere to such an extent that the atmosphere will not support the combustion of lamps or even will not support life. (4) Coals differ as regards their power of absorbing oxygen. Absorption of oxygen is never accompanied by a molecular proportional increase in the amount of carbon dioxide produced, for the oxygen combines with certain unsaturated hydrocarbon compounds in the coal. (5) The amount of water vapor in air is principally a function of the temperature of the air; hence, if air that is comparatively dry enters a mine, if the temperature of the mine is higher than the outside temperature, and if the mine is comparatively wet, then the proportion of water vapor taken up by the mine air may be greatly in excess of that in the atmospheric air.

In mines other than coal mines the chief factors that decrease the oxygen content of the mine air are the velocity of the air current, the extent of the workings traversed by the air, the amount of oxygen taken from the air, and the amount of carbon dioxide or other suffocating gases added to it by the decay of mine timbers, the burning of lights, the breath of men and of animals, and the oxidation of minerals, and also by the gases given off from the rocks penetrated.

#### COMPOSITION OF THE ATMOSPHERE.

Pure dry air as analyzed contains, by volume, 20.93 per cent of oxygen, 0.03 per cent of carbon dioxide, and 79.04 per cent of nitrogen. Included in the nitrogen content are the four inactive gases—argon, krypton, neon, and xenon. These, with the exception of argon, which constitutes 0.94 per cent of air, are present in exceedingly small proportions. Water vapor is also present in the air, varying greatly in amount at different times and in different places. The temperature of the air is the most important factor in determining water-vapor content.

Different analysts have found that pure air differs slightly in composition from the figures given above, but these variations, with the exception of carbon dioxide, for which the occasional variation may be one or two hundredths of 1 per cent, are due to unavoidable inaccuracies in the analyses. Pure air has the same composition by volume at sea level or on mountain peaks or at heights that may be reached only with balloons. This constancy of composition is brought about by a gaseous exchange between the plant and the animal life, because animals throw off carbon dioxide, whereas all the higher forms of plant life absorb it. All of the carbon dioxide of the living world comes from the 0.03 per cent of carbon dioxide in the atmosphere.



**TRANSFER OF OXYGEN AND CARBON DIOXIDE IN BREATHING.**

When a man breathes, air enters the lungs. The oxygen from the air passes through the delicate lining membrane of the air cells of the lungs into the blood, where it forms a loose chemical combination with the hæmoglobin in the red corpuscles. Thus loosely combined the oxygen is conveyed to the heart, and thence through the arteries to the capillaries, where it separates from the hæmoglobin and passes through the capillary walls to the tissues, where it is consumed. Carbon dioxide passes from the tissues in a corresponding manner, forms a loose combination (as bicarbonate and possibly other compounds) in the blood and is conveyed by the veins to the lungs, where the combinations break down, and the carbon dioxide passes from the lining membrane of the lungs into the breath.

**EFFECTS ON MAN OF VARIATIONS IN COMPOSITION AND AMOUNT OF AIR.****EFFECT OF CARBON DIOXIDE.**

As regards the amount of carbon dioxide present that may give rise to symptoms of poisoning, Haldane and Smith<sup>a</sup> carried out a number of experiments on animals and men and demonstrated that these symptoms did not begin to appear till 3 to 4 per cent of the gas was present, when the breathing became slightly affected. Men can, however, go on working for a considerable time in this atmosphere without feeling serious discomfort, although they will certainly become quickly fatigued, and great exertion will cause panting; but Haldane kept animals for weeks in this atmosphere without causing them much inconvenience. A proportion of 7 to 8 per cent causes more apparent symptoms, and with 10 per cent the distress is great, the headache becomes much more severe, there is marked dyspnœa, throbbing pulse, and flushing of the face, and the gas begins to have a stupefying effect. With 12 to 15 per cent, cerebral symptoms appear, and the patient soon becomes unconscious. Death may take place after exposure for several hours to 25 per cent, but Haldane found that some animals may breathe a much greater percentage, even 50 per cent, without dying.

The concentration of carbon dioxide in the lungs is automatically regulated, so that it is remarkably constant, practically without relation to changes in the rate of breathing, provided respiration is not

<sup>a</sup> Haldane, J. S., and Smith, Lorain, Physiological effects of air vitiated by respiration: Jour. Path. Bact., vol. 1, 1892, p. 174.

forced. This relation was shown by experiments of Haldane and Priestly,<sup>a</sup> the results of which follow:

*Results of experiments of Haldane and Priestly to determine relation between rapidity of respiration and concentration of carbon dioxide in lungs.*

Subject.	Respirations per minute.	Proportion of carbon dioxide in alveolar air—		
		At end of inspiration.	At end of expiration.	Mean.
		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
J. S. Haldane.....	9	5.59	5.87	5.73
Do.....	19	5.56	5.70	5.63
Do.....	9	5.33	5.47	5.40
Do.....	20	5.44	5.60	5.52
J. G. Priestly.....	10.5	5.95	6.74	6.35
Do.....	30	5.98	6.05	6.02

Even during muscular exertion Haldane<sup>a</sup> and Priestly found that when the percentage of carbon dioxide in the inspired air was increased to 4 or 5 per cent the partial pressure of the carbon dioxide in the air in the air cells of the lungs was practically constant.

The effect of breathing air containing carbon dioxide is to stimulate the so-called respiratory center in the brain and increase the depth of breathing. This increase in the depth of breathing is brought about by the smallest increase in the percentage of carbon dioxide in the inspired air, and is sufficient to keep constant the percentage of carbon dioxide in the alveoli, or air cells, of the lungs. This response of the respiratory center of the brain to carbon dioxide is seen during muscular work. The latter increases the production of carbon dioxide in the arterial blood, resulting in an involuntarily increased depth of breathing proportionate to the extra production of carbon dioxide, hence there results the rapid breathing (hyperpnoea) seen during exercise. So small an amount of carbon dioxide in the atmosphere as 1 or 2 per cent does not materially endanger the life and comfort of those who breathe it, but does decrease their efficiency as workmen. A man is forced to breathe a larger amount of air in a given time, and this breathing consumes energy just as the work he is doing consumes energy.

An increase of 50 per cent in the amount of air breathed is brought about by the presence of 1.4 per cent CO<sub>2</sub> in the inspired air, according to Douglas, Haldane, Henderson, and Schneider.<sup>b</sup> They add that panting then becomes excessive even if a man does a great deal less muscular work than he could do in normal air.

<sup>a</sup> Haldane, J. S., and Priestly, J. G., The regulation of the lung ventilation: Jour. Physiol., vol. 32, 1905, p. 225.

<sup>b</sup> Douglas, C. G., Haldane, J. S., Henderson, Y., and Schneider, E. C., Physiological observations made on Pikes Peak, Colo.: Philos. Trans. Roy. Soc. London, ser. B, vol. 203, February, 1913, pp. 220-221.



## EFFECT OF DIMINISHED OXYGEN SUPPLY.

Abnormally rapid breathing is also brought about by want of oxygen. In some individuals such breathing is produced when the oxygen percentage of the air is less than 13 (normally about 21) per cent. Rapid breathing is produced much more quickly by excess of carbon dioxide than by a corresponding deficiency of oxygen. The important practical point to remember is that rapid breathing caused by carbon dioxide starts long before danger becomes serious, whereas when it is produced as the result of a deficiency of oxygen it is a grave symptom and points to serious danger.

Bert<sup>a</sup> showed that the abnormal symptoms and dangers associated with low barometric pressure depend, not on the diminished mechanical pressure, but on the diminished partial pressure of the oxygen and consequent imperfect aeration of the arterial blood with oxygen. He found that definite symptoms of want of oxygen began to appear whenever the partial pressure of the oxygen was reduced below a certain limit. Thus at ordinary atmospheric pressure a cat died when the proportion of oxygen was reduced to about 4.5 per cent, or a partial pressure of 4.5 per cent of an atmosphere. At a barometric pressure of 0.5 atmosphere, on the other hand, the animal died when the oxygen was reduced to about 9 per cent, which is again 4.5 per cent of an atmosphere, and at a pressure of 2 atmospheres a reduction of the oxygen percentage to 2.25 per cent was needed. He established the general law that the physiological action of a gas depends on its partial pressure, and further showed that the percentage by volume of oxygen taken up by the blood either in the lungs or outside the body depends on the partial pressure of the oxygen, and is greatly reduced when the lowering of oxygen pressure becomes dangerous.

When men or animals, even though at rest, are subjected to atmospheres deficient in oxygen or at low pressures, marked hyperpnœa is rapidly produced. Marked hyperpnœa caused by oxygen want is only temporary, and is due to the fact that the want aids carbon dioxide in exciting the respiratory center.<sup>b</sup> For a short time the proportion of carbon dioxide already present in the blood and tissues is more than enough to excite the center. When the excess has been eliminated by a temporary hyperpnœa the breathing again becomes quiet, and if the transition to want of oxygen is gradual the temporary marked hyperpnœa is not noticed. Want of oxygen without the aid of carbon dioxide does not excite the respiratory center at all.<sup>c</sup>

<sup>a</sup> Bert, Paul, *La pression barométrique*, 1878.

<sup>b</sup> Haldane, J. S., and Poulton, E. P., Effects of want of oxygen on respiration: *Jour. Physiol.*, vol. 37, 1908, p. 390.

<sup>c</sup> Henderson, Yandell, Apnoea and shock: *Am. Jour. Physiol.*, vol. 25, 1910, pp. 310, 385.

## EFFECT OF AIR AT HIGH ALTITUDES.

Men ascend to high altitudes where the air is "thin," and consequently the partial pressure of the oxygen as compared with that at sea level is small. Under the same conditions of temperature and pressure the expansion of different gases is practically the same. Hence the proportion by volume of oxygen, nitrogen, etc., in air remains the same, no matter what the altitude, but the percentage by weight varies as the pressure changes. Air at sea level and at a barometric pressure of 760 mm. contains, by weight, about 23 per cent of oxygen, but at half that pressure, or 380 mm., contains only half of 23 per cent, or 11.5 per cent.

The table following shows the barometric pressures at different heights, also the partial pressure of the oxygen—that is, the percentage of oxygen that would correspond to the same attenuation of air at sea level.

*Partial pressures of oxygen corresponding to different barometric pressures.*

Altitude.		Inches of mercury.	Barometric pressure, mm. of Hg.	Partial pressure of oxygen compared to pressure at sea level.
Meters.	Feet.			
0	0	30.0	760	21
1,000	3,280	26.0	670	18.5
2,000	6,560	23.4	592	16.3
3,000	9,840	20.6	522	14.4
4,000	12,290	18.2	460	12.7
5,000	16,405	16.0	406	11.2
6,000	19,580	15.4	358	9.9
7,000	22,965	12.5	316	8.7
8,000	26,245	11.0	279	7.7

In explorations of the Himalayas the Duc d'Abruzzi and his companions climbed to a height of 24,580 feet, the barometric pressure being 312 mm., and the partial pressure of the oxygen about 8 per cent.<sup>a</sup> At this point the explorers felt no discomfort during rest and seemingly did not experience any difficulty in performing the work required in climbing.

However, after the explorers had lived for several weeks at a height of 17,000 feet above sea level, the atmosphere did work some harmful effects, revealed only gradually in a slow decrease of appetite and consequent lack of nourishment, but without any disturbance of digestion. Of course, ultimately this insufficient nourishment would have caused a lowering of vitality, loss of flesh, and a certain amount of anæmia. However, the effect was so slow that even at the end of two months the men were able to make long marches without becoming excessively tired.

<sup>a</sup> Filippi, Di Filippo de, Karakoram and western Himalaya, 1909, pp. 361-368.



Many people are affected at a height of 6,000 to 9,000 feet. Every-body suffers from shortness of breath and fatigue at 12,000 feet, and serious symptoms frequently develop at 16,000 feet. At this height the partial pressure of the oxygen corresponds to air at sea level containing about 11.5 per cent oxygen. However, people can live at very high altitudes. In Peru, Bolivia, and northern Chile a large proportion of the population live above 10,000 feet. The elevation of the observatory at El Mista, in the Andes, is about 19,000 feet. It takes time, however, for the body to adapt itself to such an attenuation of the air. A person unaccustomed to it and suddenly plunged into such an atmosphere would experience severe distress.

#### EFFECT OF BREATHING A DECREASING OXYGEN SUPPLY.

Some instructive experiments on this point, in which one of the authors participated, are described in Technical Paper 122<sup>a</sup> and summarized in Technical Paper 109.<sup>b</sup> The experiments were made under the direction of Yandell Henderson, professor of physiology at Yale University.

A man breathed air in and out of a bag having a capacity of about 70 liters. By means of a can of caustic potash inserted between the man's mouth and the bag the exhaled carbon dioxide was removed. Under these conditions, of course, the oxygen content of the air breathed gradually fell. When it had fallen to about 7 per cent the subject lost consciousness for a few seconds.

The mode of action of the "oxygen want" is instructive. The subject felt no urgent warning symptoms up to the time of collapse. In fact, he wanted to continue, and just before collapse made a show of resistance against discontinuing the experiment. He felt no real distress until some time after the experiment. The next day he was decidedly unwell.

#### EFFECT OF A MINE ATMOSPHERE LOW IN OXYGEN.

In its insidious action "oxygen want" acts as carbon monoxide frequently does, that is, when the oxygen is slowly decreased. One difficulty in comparing the mode of action of the two lies in the scarcity of experimental data regarding the effects on men of atmospheres containing small percentages of oxygen.

Much light is thrown on the action of the two gases by an accident that occurred in the Lodge Mill Colliery, Huddersfield, England.<sup>c</sup>

<sup>a</sup> Burrell, G. A., and Oberfell, G. G., Effects of atmospheres deficient in oxygen on small animals and on men: Tech. Paper 122, Bureau of Mines, 1915, pp. 7-9.

<sup>b</sup> Burrell, G. A., and Oberfell, G. G., Composition of natural gas used in 25 cities, with a discussion of the properties of natural gas: Tech. Paper 109, Bureau of Mines, 1915, pp. 16-18.

<sup>c</sup> Lloyd, W. D., The use of rescue apparatus at Lodge Mill Colliery, Huddersfield: Coll. Guard., vol. 106, Nov. 7, 1913, p. 957.

Three men were in a disused part of the mine, and two of them, A and B, were overcome by the black damp at 3.30 a. m. A third, C, was overcome at 4 a. m. in an attempt to rescue his comrades. At 1 p. m. of the same day rescuers equipped with breathing apparatus removed C and about an hour later removed A. A little later B was found dead. Both A and C were in a critical condition but still breathing. A died three days later. The air in which the men were overcome contained a high percentage of methane and had a correspondingly low oxygen content. Haldane,<sup>a</sup> in speaking of this disaster, made the following comment:

All the facts recorded indicate that the men were overcome by the insufficiency in the oxygen percentage of the air. There was no reason to suspect the presence of carbon monoxide as there was no gob fire or heating. In addition, the blood of the dead man was black, not red, as it would have been if death had been due to carbon monoxide. It is probable that sufficient fire damp was present to reduce the oxygen to 7 or 8 per cent. The fact that A did not recover was due to exactly the same cause which often prevents men recovering after severe carbon monoxide poisoning. The tissues have been severely damaged by the prolonged exposure to dearth of oxygen, so that, although the oxygen supply is completely restored, recovery is doubtful. In the case of A the post-mortem examination revealed the fact that the heart was dilated. Probably the heart muscles and the other tissues were in a condition of fatty degeneration caused by the want of oxygen. The writer has seen other similar cases of dilatation simulating severe heart disease, and only slowly recovering after prolonged exposure to carbon monoxide poisoning. So far as he is aware, however, this is the only recorded case of death, after partial recovery, from exposure to an atmosphere which was simply deficient in oxygen, apart from the presence of carbon monoxide.

#### EFFECT OF ATMOSPHERES LOW IN OXYGEN ON CANARIES AND MICE.

Some experiments to determine the effect of atmospheres low in oxygen on mice and canaries, performed by the authors of this bulletin, are described in Technical Paper 122.<sup>b</sup> In conducting the experiments, atmospheres containing various percentages of nitrogen and oxygen were prepared in 10-liter bell jars. For the nitrogen supply a tank containing nitrogen with a small percentage of oxygen was obtained. The atmosphere in the tank analyzed 97.8 per cent nitrogen and 2.2 per cent oxygen.

In the experiments with canaries three birds, designated A, B, and C, were used. Tabulated results of the experiments follow:

---

<sup>a</sup> Haldane, J. S., Note on paper by W. D. Lloyd: Coll. Guard., vol. 106, Nov. 7, 1913, pp. 957-958.

<sup>b</sup> Burrell, G. A., and Oberfell, G. G., Effect of atmospheres deficient in oxygen on small animals and on men: Tech. Paper 122, 1915, pp. 5-6.



TABLE 1.—*Results of experiments to determine effect on canaries of breathing atmospheres low in oxygen.*

Trial No.	Canary.	Composition of atmosphere during experiment.						Effect on canary.
		At beginning.			At end.			
		Oxy-gen.	Nitro-gen.	Carbon dioxide.	Oxy-gen.	Nitro-gen.	Carbon dioxide.	
1	B.....	<i>P. ct.</i> 9.42	<i>P. ct.</i> 90.48	<i>P. ct.</i> 0.10	<i>P. ct.</i> 8.50	<i>P. ct.</i> 90.90	<i>P. ct.</i> 0.60	Immediate distress evinced by rapid breathing, open bill, and tendency to wabble. In 10 minutes the bird was seemingly in normal condition except for slightly increased rate of breathing. It was removed from the atmosphere after 1 hour.
2	C.....	9.25	90.65	.10	.....	.....	.....	Behaved similarly to canary B in trial 1.
3	A.....	7.65	92.25	.10	7.55	92.05	.40	Showed immediate distress by panting and unsteadiness, but did not collapse. It did not evince any more distress than canary B in trial 1. It was left in the atmosphere for 1 hour.
4	C.....	7.83	92.07	.10	.....	.....	.....	Collapsed as soon as it was placed in the atmosphere. It breathed very slowly, with eyes and bill closed, but recovered its normal state in less than a minute after it had been removed to fresh air.
5	A.....	7.10	92.80	.10	.....	.....	.....	Showed distress, but did not collapse in 20 minutes. When canary C was placed in the atmosphere it immediately collapsed.

The next experiments were made with white mice instead of canaries. Tabulated results follow.

TABLE 2.—*Results of experiments to determine effect on white mice of atmospheres deficient in oxygen.*

Trial No.	Composition of atmosphere during experiment.						Effect on mouse.
	At beginning.			At end.			
	Oxy-gen.	Nitro-gen.	Carbon dioxide.	Oxy-gen.	Nitro-gen.	Carbon dioxide.	
1	<i>P. ct.</i> 7.07	<i>P. ct.</i> 92.83	<i>P. ct.</i> 0.10	<i>P. ct.</i> 7.75	<i>P. ct.</i> 92.05	<i>P. ct.</i> 0.20	Showed immediate symptoms of distress. At times it appeared normal and again appeared sluggish. Did not collapse. It was left in the atmosphere 1 hour. Same as on mouse in trial 1. Same as on mice in trials 1 and 2. Showed immediate distress and collapsed in 6 minutes. One minute after removal from the atmosphere it was on its feet.
2	6.89	93.01	.10	7.12	92.78	.10	
3	6.70	93.00	.30	-----	-----	-----	
4	5.90	94.00	.10	-----	-----	-----	

## GENERAL COMMENTS ON TESTS.

Canaries show some distress in atmospheres containing 9.4 per cent oxygen. The distress is more pronounced in atmospheres containing as low as 7.6 or 7.8 per cent, but even in an atmosphere containing as little as 7.1 per cent of oxygen they may or may not collapse. Mice are slightly more resistant to atmospheres low in oxygen. In an atmosphere containing about 7 per cent of oxygen they show some distress and become more or less sluggish if left in it. One mouse resisted an atmosphere containing as little as 5.9 per cent of oxygen before it collapsed. With that percentage, collapse did not occur until the mouse had been exposed for 6 minutes.

**EFFECTS OF OXYGEN DEFICIENCY AS SHOWING RELATIVE VALUE OF BIRDS AND MICE IN EXPLORING MINES.**

From the foregoing experiments on birds, mice, and men, it will be seen that oxygen want begins to affect men in about the same time as it does mice and canaries, and the percentages that cause collapse are not much different. Animals differ slightly as to their resistance to collapse in atmospheres low in oxygen, and presumably the same is true of men. The fact is evident that exploring parties in mines can not satisfactorily use birds and mice to detect atmospheres low in oxygen.

Canaries are of chief value in showing by their behavior the presence of carbon monoxide. Afterdamp may contain enough oxygen, about 17 per cent, to support the flame of an oil-fed lamp, so that exploring parties entering it would have no means of knowing that the air is low in oxygen, although the presence of dangerous percentages of carbon monoxide could be detected by the action of the canaries, even if it is not shown by the flame of the safety lamp. When mine-exploring parties, unequipped with breathing apparatus, reach an atmosphere in which oil lamps are extinguished, they are likely to proceed close to or into dangerous atmospheres if they advance farther, even if provided with canaries for detecting vitiated air. The acetylene lamp does not become extinguished until the proportion of oxygen falls to about 13 per cent. In such an atmosphere a party is not in immediate danger, although close at hand there may be atmospheres considerably lower in oxygen, and hence dangerous, into which a person might walk in a very short time. However, any party exploring a mine after a disaster and using canaries would also be equipped with oil safety lamps and with electric lamps, so that acetylene lamps need hardly be considered.



**EFFECT OF TEMPERATURE AND MOISTURE ON MAN.**

The temperature of the atmosphere is important in that it governs the capacity of air for absorbing moisture. A hot, moist atmosphere prevents the evaporation of perspiration from the body. If, in addition, hot, moist air is stationary, it becomes entangled between the clothing and skin, and becomes warmed to body temperature. Hence, the body can not lose heat to the air, and the skin becomes warmed and bathed in perspiration. Putting a fan in operation whirls the hot, warm air away from the body and allows cooler air to take its place, accounting for the beneficial effect of a fan in a room, even if fresh outside air is not finding access. If the wet-bulb temperature is 99° F. (body temperature), the body will not lose heat, but if the external temperature is 79° or 69° F., perspiration can evaporate. Experience has taught that when the wet-bulb temperature exceeds 75° F. the amount of work that a man can do begins to fall off. Ventilating engineers in this country try to arrange ventilating equipment so that a wet-bulb temperature of 72° F. will not be exceeded. They aim to maintain a relative humidity varying between 30 and 80 per cent. The standard for dry-bulb temperature is about 68° F.

Cadman and Whalley,<sup>a</sup> in an investigation of ventilation in English coal mines, observed that at a wet-bulb temperature of 72° F. heavy clothing was removed and only light clothing worn. At 80° F. hard work was possible, provided the maximum body surface was exposed, the capacity for work being greatly increased, however, if a current of air or a breeze passed over the body. With wet-bulb temperatures of from 80° to 85° F. work was seriously affected, and hard work was almost impossible.

**VENTILATION OF BUILDINGS ABOVE GROUND.**

In designing the ventilation systems of public buildings, houses, and other places where people meet, the effects of carbon dioxide and oxygen are subordinated to other conditions, such as temperature, humidity, and keeping the air moving. At sea level, in the most ill-ventilated room, the weight of the oxygen in a cubic foot of the air never is as low as in the open air at high altitudes, and it is seldom that the carbon dioxide content rises above 0.10 or 0.20 per cent, although the air in such a room may be unfit to breathe.

The ventilation in rooms is not essentially a matter of correcting diminished oxygen or increased carbon dioxide but of keeping the temperature, the relative moisture, and the movement of the air in a proper state. However, in the ventilation of rooms where people

<sup>a</sup> Cadman, John, and Whalley, E. B., Report of an inquiry into the ventilation of coal mines and the methods of examining for fire damp: Royal Commission on Mines, England, 1909, p. 4.

congregate the carbon dioxide and the oxygen tests may be excellent criterions of the state of ventilation. Haldane and Osborne<sup>a</sup> have investigated the ventilation of a large number of workshops and factories and have recommended that the standard of ventilation be such that the proportion of carbon dioxide does not exceed 0.12 per cent during daylight or 0.20 per cent after dark when oil or gas is used for lighting. The carbon dioxide percentage is specified because its determination is easy and its presence a sign that there is no free ventilation and that the air tends to become humid and warm. However, the vitiation of air in overcrowded rooms may arise from the products of disease, want of cleanliness of the occupants, or the room itself. In fact, air that, judged by the carbon dioxide standard, is sufficiently pure, may be exceedingly impure when judged by the number of microorganisms in it, and vice versa. The test of smell, of feeling of comfort or discomfort, in breathing the air of a room may give results equally at variance with the carbonic acid test. Some people are more influenced by odor, others by an increased quantity of carbon dioxide or moisture or an increased temperature.

Men who habitually work in bad air that is not sufficiently tainted to cause acute symptoms may suffer from slight anæmia, with disinclination for work, rapid breathing on exertion, lassitude, loss of appetite, and other symptoms of indigestion. Return to work in better air, however, soon puts them right.

#### VENTILATION CONDITIONS IN MINES.

The foregoing discussion of the factors to be considered in ventilation studies has largely to do with conditions aboveground and in public halls, houses, and other places where people congregate. With modifications it is applicable to conditions in coal and in other mines. In meetinghouses aboveground the carbon dioxide in excess of the proportion found in atmospheric air is principally derived from exhaled air, and if present, even in proportions as high as 0.20 per cent, is usually a sign of stagnant air and poor ventilation. It has been found that this proportion of carbon dioxide usually accompanies very bad ventilation in that the air is stagnant and oppressive, with perhaps high temperature and the odor of perspiring bodies.

In coal mines carbon dioxide arises principally from the action of the air on the coal, and 0.2 to 0.3 per cent is frequently found in the cool, swiftly moving air of returns, where 50,000 or more cubic feet of air is passing per minute and the wet-bulb or dry-bulb temperature does not exceed 65° F. The relative humidity may be high, almost 100 per cent in many mines, owing to artificial methods of watering to allay coal dust, but where the temperatures are not high

<sup>a</sup> Haldane, J. S., and Osborne, O. O., Ventilation of factories and workshops: English Govt. Blue Book, 1902.

the high humidity has no bad effect. Hence a proportion of carbon dioxide that would be easily tolerated in a coal mine might be indicative of extremely poor ventilation in the room of a house.

A similar statement applies to oxygen. The oxygen content is scarcely ever normal in a coal mine, owing to the absorption of oxygen by the coal; in fact, a diminution of 1 per cent is not uncommon. Such a diminution in an assembly room would usually be accompanied by intolerable conditions of ventilation. However, the proportion of oxygen can not be diminished too much even in mines. It is the author's experience that as little as 19 per cent of oxygen is rarely found in the moving air of coal mines. However, at some of the working faces of poorly ventilated mines only 17 per cent of oxygen is found.

In mines other than coal mines, carbon dioxide is given off by the burning of lights, by the decay of mine timbers, and, in some mines, by the rocks penetrated. At poorly ventilated working places in some metal mines the carbon dioxide content may be dangerously high or the oxygen content dangerously low, or both conditions may prevail.

As indicated above, an oxygen content as low as 17 per cent is itself without harm, for people live at altitudes where the proportion of oxygen corresponds to this percentage at sea level. The air of many health resorts contains even less oxygen by weight than does the air of Denver, but the air at those resorts is fresh, moving, and invigorating, and usually rather dry. Mine air that contains so small a percentage of oxygen is in no sense comparable to the air of these health resorts. If the air in a part of a certain mine contains only 17 per cent by volume of oxygen not enough fresh air is finding access there; that is, the air is more or less stagnant, perhaps with all the attendant evils discussed previously.

The difference between ventilation conditions in buildings above-ground and in coal mines is in the degree of vitiation of the air as regards low oxygen and high carbon dioxide; aboveground relatively small oxygen and carbon dioxide changes—a few tenths of 1 per cent—usually represent bad air; belowground these small percentages are insignificant. In England, the law requires that the oxygen content shall not fall below 19 per cent or the carbon dioxide content rise above 1.25 per cent in any part of the mine.<sup>a</sup> Mines in this country can easily meet these requirements.

The amount of fresh air per man introduced into coal mines far exceeds that allotted by architects and engineers for ventilating public buildings. As a basis on which to calculate the air necessary for proper ventilation of public buildings, one may consider the

---

<sup>a</sup> Coal mines act, Great Britain, 1911.



carbon dioxide produced by an adult as 0.6 cubic foot per hour. Hence, in 6,000 cubic feet of air, he will produce per hour a pollution amounting to 1 part of carbon dioxide in 10,000 parts of air. If the excess of carbon dioxide were to be kept down to this figure it would be necessary to supply 6,000 cubic feet of fresh air per hour; if the permissible excess was to be 2 parts in 10,000, half this supply would suffice, and so on. The amount of fresh air allowed by ventilating engineers in this country varies from 1,800 cubic feet per hour per person for school buildings, auditoriums, theaters, factories, etc., to 12,000 cubic feet per hour per person in surgical and contagious hospitals.

In the nongaseous bituminous mines of Pennsylvania the minimum quantity of air per man is 150 cubic feet per minute, or 9,000 cubic feet per hour. In a mine where explosive gas is generated in dangerous quantities the minimum is 12,000 cubic feet per hour per man. These quantities exceed the quantity stipulated in the ventilation of large public halls. Consequently in the moving airways of such mines the air is fresh and wholesome. The temperature is seldom high, and usually bad ventilation exists only at some working faces where the air does not find ready access.

There has been considerable discussion regarding a scheme proposed in England to lower the oxygen content of the air in mines to a point (about 17 per cent) where the risk from explosion of mixtures of methane and air is reduced. Most mining men are opposed to such procedure on physiological grounds. The author's experience has indicated that the oxygen content has to be reduced below this figure to lessen appreciably the liability of gas explosions. (See pages 24-25.)

Some investigators have argued that disastrous coal-dust explosions of recent years can be attributed largely to the better ventilation of mines at present as compared with the ventilation found years ago. To make such a comparison just involves the consideration of too many factors to warrant discussion here. There is truth in the statement, at least as regards mixtures of methane and air, to the extent that greater violence is exerted by an explosive mixture if the latter has a high initial velocity of its own. The authors have seen this demonstrated many times in an explosion gallery of their own construction.<sup>a</sup> But an unbiased consideration of evidence accumulated through long experience must lead one to accept the necessity of plenty of fresh air. Whether for a miner working underground, a man at a health resort, or an assemblage in a large public building, all the fresh air that can reasonably be obtained is needed.

<sup>a</sup> Details to be published later. The high initial velocities of the gallery tests are, however, probably almost unknown in mines where an explosive mixture of methane and air might exist.

## ACTION OF OXYGEN ON COAL.

Many investigations have been made into the mode of absorption of oxygen by coal, which is of much practical importance, for this reaction is mainly responsible for the depletion of oxygen and increase of carbon dioxide in the atmosphere of a coal mine. The detrimental effect is that in some parts of mines atmospheres are produced so low in oxygen that they will not support the combustion of lights or that they impair the health of the miners. By sealing off affected areas, oxygen absorption by coal can be utilized to extinguish mine fires, to prevent spontaneous combustion, or to produce an atmosphere so low in oxygen that explosions of methane and air, or coal dust and air, can not occur.

The exact manner in which coal absorbs oxygen is not clearly understood. A solid substance can hold gases actually dissolved in its interior. With coal, the absorption of oxygen is complicated by the fact that, in addition to this physical action, there is also a very complex chemical action. Part of the oxygen is converted into water, part into carbon dioxide, and part is retained as combined oxygen, producing compounds richer in oxygen than the coal itself. Part of the carbon dioxide is retained by the coal, for coal at ordinary temperatures has an enormous capacity for holding carbon dioxide.

Many investigations have been made into the absorption and liberation of gases by coal, as is indicated by the following selected bibliography:

*Selected bibliography on absorption and liberation of gases by coal.*

- ANDERSON, W. C. A contribution to the chemistry of coal, with special reference to the coals of the Clyde Basin. Phil. Soc. Glasgow, vol. 29, 1897, pp. 72-96.
- BEDSON, P. P. The gases inclosed in coal dust. Chem. News, vol. 68, 1893, p. 187.
- BOUDOUARD, M. O. Études sur les charbons, produits resultant de leur oxydation. Bull. Soc. chim. France, ser. 4, t. 5, 1909, pp. 377-380.
- CHAMBERLAIN, R. T. Notes on explosive mine gases and dusts, with special reference to the explosions in the Monongah, Darr, and Naomi coal mines. Bull. 26, Bureau of Mines, p. 32 (reprint of U. S. Geol. Surv. Bull. 383).
- HARGER, JOHN. The detection of gob fires. Trans. Inst. Min. Eng., vol. 46, 1913-14, p. 370.
- LAMPLUGH, F. E. E., and HILL, A. M. The slow combustion of coal and its thermal value. Trans. Inst. Min. Eng., vol. 45, 1913, p. 629.
- PARR, S. W., and WHEELER, W. F. Year book of Illinois State Geological Survey. Ill. State Geol. Survey Bull. 8, 1907, p. 168.
- TAFFANEL, J. De l'alteration des poussières de houille exposées a l'air. Original Communications, 8th Int. Cong. Appl. Chem., vol. 10, 1912, p. 277.
- THRELFALL, RICHARD. The spontaneous heating of coal, particularly during shipment. Jour. Soc. Chem. Ind., vol. 38, 1909, p. 759.
- TROBRIDGE, F. G. Gases inclosed in coal and certain coal dusts. Jour. Soc. Chem. Ind., vol. 25, 1906, p. 1129.

Recent investigations into the action of air on coal are those of Winmill and Graham.<sup>a</sup>

Winmill<sup>b</sup> determined that under his conditions of experiment there was little difference between the rates of oxidation by air, except for mother of coal and slate, between various parts of a seam of coal that he sampled. Hard coal, soft coal, cannel coal, jacks, shale, and mother of coal were tried. He also found that the amount of oxygen absorbed was roughly proportional to the amount of carbonaceous matter present in each substance. One hundred and fifty grams of 200-mesh coal absorbed 169 to 830 c. c. of oxygen at a temperature of 30° C. Winmill states that there are two factors in the oxidation of coal—a rapid first reaction, which is soon over, and a slow reaction, responsible for the major part of the oxygen absorption. He found, as have other investigators, that oxidation of coal, so rapid that the coal heats up, can take place in the absence of pyrites, only a small amount of carbon dioxide being produced. From the hard coal at 30° C. less than 20 c. c. of carbon dioxide was formed during the absorption of 20 c. c. of oxygen. A reduction of the proportion of oxygen to 8.28 per cent (not of the quantity of air) did not slow up the reaction very perceptibly.

With 100-mesh dust the rate of absorption was 72 per cent of that with 200-mesh dust. Increasing the temperature of the experiment from 40° to 60° C. practically doubled the rate of absorption. In another communication Winmill<sup>c</sup> gives the results of some experiments on the rate of absorption of carbon dioxide by coal. The temperature of the experiment was 30° C. The proportion of carbon dioxide absorbed by 100 grams of coal varied from 0.97 c. c., with 0.10 per cent of carbon dioxide in the atmosphere, up to 472 c. c., with an atmosphere containing 100 per cent of carbon dioxide.

#### INFLUENCE OF MOISTURE ON RATE OF ABSORPTION OF OXYGEN BY COAL.

Fayol<sup>d</sup> maintains that there is little evidence in support of the view that moisture accelerates the firing of coal heaps.

Richters<sup>e</sup> concludes that more oxygen is absorbed by dry than by moist coal.

Mahler<sup>f</sup> found that coal in the presence of oxygen under pressure absorbed a considerably larger quantity when moist than when dry.

<sup>a</sup> Graham, J. I., The absorption of oxygen by coal: *Trans. Inst. Min. Eng.*, vol. 59, Feb., 1915, p. 35.

<sup>b</sup> Winmill, T. F., The absorption of oxygen by coal: *Trans. Inst. Min. Eng.*, vol. 46, 1913-14, pp. 563, 568.

<sup>c</sup> Winmill, T. F., *Op. cit.*, vol. 48, Feb., 1915, p. 514.

<sup>d</sup> Threlfall, Richard, The spontaneous heating of coal, particularly during shipment: *Jour. Soc. Chem. Ind.*, 1909, vol. 28, p. 763.

<sup>e</sup> Threlfall, Richard, *Op. cit.*, p. 759.

<sup>f</sup> Mahler, M. P., Experiments on the oxidation of coal: *Coll. Guard.*, 1913, vol. 106, p. 891.



Graham<sup>a</sup> determined that (1) at temperatures below 50° C. coal dust when moist absorbs oxygen at a rate approximately half as great again as when dry; (2) that coal dust heated for some hours at a temperature of 100° C. in a vacuum during subsequent oxidation absorbs oxygen at the same rate as when not previously heated; (3) that the absorption of oxygen by coal is independent of bacterial activity.

Porter and Ovitz<sup>b</sup> found that 10 kilograms of coal (22 pounds) in a bottle absorbed during the first day after mining nearly half the oxygen from 10 liters of air and gave off little more than one-tenth as much carbon dioxide as would have been formed if all the oxygen had combined with carbon to produce carbon dioxide. Four coals that Porter and Ovitz tested in bottles absorbed in 7, 9, 10, and 17 months 4.362, 2.853, 4.115, and 6.935 volumes of oxygen, as compared with the volumes of the coals. From time to time the gas that accumulated in each bottle was drawn off in order to relieve the pressure and permit the inflow of oxygen.

Porter and Ralston<sup>c</sup> found that with the same coal the rate of oxidation increased from 0.21 c. c. of oxygen absorbed per gram of dry coal at 40° C. to 27.68 c. c. absorbed at 200° C., the duration of each test being 60 minutes. They found that the oxygen content of the air must be reduced below 15 per cent, or to a pressure of about 100 mm. when pure oxygen is used, before the effect on the rate of oxidation becomes serious, and that the presence of as much as 10 per cent of carbon dioxide in the atmosphere has no retarding influence on the rate of oxidation at 200° C.

Winmill, it will be remembered, found that if the percentage of oxygen in air be reduced to 8.20 the reaction did not slow perceptibly, but his results are not in harmony with those of Porter and Ralston. It is certain, of course, that the rate of oxidation must eventually fall decidedly when the oxygen content of the air becomes low, and must be zero when the oxygen content of the atmosphere is zero.

#### ABSORPTION OF OXYGEN AND PRODUCTION OF CARBON DIOXIDE FROM ATMOSPHERIC AIR BY WOOD.

Many metal mines are heavily timbered and in some mines, especially in dead ends where the air is stagnant, the wood is moist, and much timber is present, the carbon dioxide content of the air is increased and the oxygen content is decreased.

<sup>a</sup> Graham, J. I., The absorption of oxygen by coal: Trans. Inst. Min. Eng., vol. 59, Feb., 1915, p. 35.

<sup>b</sup> Porter, H. C., and Ovitz, F. K., The escape of gas from coal: Tech. Paper 2, Bureau of Mines, 1911, pp. 6-7.

<sup>c</sup> Porter, H. C., and Ralston, O. C., A study of the oxidation of coal: Tech. Paper 65, Bureau of Mines, 1914, p. 13.

Some results by the authors on the action of air on wood are given in Table 3 following. Dried and seasoned pieces of wood plank were sawed or shaved to produce sawdust or shavings. The material was placed in bottles having a capacity of 2.5 liters and the bottles were securely closed. Through each bottle stopper was placed a glass tube provided with a stopcock to permit drawing out samples of air for analysis.

*Results of experiments to determine effect of wood in changing composition of atmospheric air.*

Kind of wood.	Weight.	Constituent.	First analysis of residual air, June 27, 1913.	Second analysis of residual air, July 15, 1913.	Third analysis of residual air, July 31, 1913. <sup>a</sup>	Fourth analysis of residual air, Aug. 21, 1913.	Fifth analysis of residual air, Sept. 16, 1913.	Sixth analysis of residual air, Dec. 12, 1913.
	Grams.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Ash sawdust.....	5	(CO <sub>2</sub> .....	0.0	0.15	3.38	11.6	18.9	20.6
		(O <sub>2</sub> .....	20.8	20.8	17.50	8.6	1.7	.0
Cypress shavings.....	76	(CO <sub>2</sub> .....	.0	.15	1.98	4.0	6.0	10.0
		(O <sub>2</sub> .....	20.9	20.80	18.75	16.7	14.4	10.1
Hemlock sawdust....	135.5	(CO <sub>2</sub> .....	.20	.20	7.87	16.6	18.0	17.7
		(O <sub>2</sub> .....	20.8	20.80	12.00	2.0	.7	.3
Oak sawdust.....	136	(CO <sub>2</sub> .....	.2	.2	8.27	17.0	19.2	20.2
		(O <sub>2</sub> .....	20.7	20.8	15.05	3.1	1.2	.4

<sup>a</sup> Water was added to each bottle 15 days after the tests were started and just prior to these analyses.

#### DISCUSSION OF RESULTS.

The experiments were started on June 6, 1913, and the first analyses of the residual air were made on June 27, 1913. No appreciable change in the oxygen or carbon dioxide content of the residual air over the composition of ordinary atmospheric air was noticed. The same statement held true of results obtained on July 15, 1913, or 39 days after the experiments were started. Water was then added to each bottle in sufficient quantity to perceptibly moisten the sawdust and shavings, and analyses were again made. After 16 days from the time the wood had been moistened, analyses were made of the residual air, and a marked increase in carbon dioxide and a marked decrease in oxygen were found. These changes continued in the sawdust samples until the oxygen had practically all disappeared. The rate of absorption of the oxygen by the fine cypress shavings was slower, probably owing to the fact that the cypress wood was not in as fine a state of division as the other woods tried.

An interesting feature of the results was that the carbon dioxide was only slightly less than the molecular equivalent of the oxygen consumed. This result differs from that obtained when oxygen is absorbed by coal at ordinary temperatures. The amount of carbon dioxide produced by coal is invariably less than the molecular equivalent of the consumed oxygen. In fact, the oxygen may entirely or

almost entirely disappear, and there may be present in the residual air only 2 or 3 per cent of carbon dioxide. Bacterial action apparently does not enter into the phenomenon, whereas as regards the reaction between wood and oxygen, bacterial action is probably almost wholly responsible.

#### EFFECT OF LOWERING OXYGEN AND RAISING CARBON DIOXIDE CONTENT ON EXPLOSIBILITY OF METHANE-AIR MIXTURES.

In coal mines, especially if the air is not in rapid circulation, the composition of the atmosphere may change quickly. The principal changes that affect the explosibility of mine atmospheres are those caused by the absorption of oxygen by the coal and the oxidation of coal to carbon dioxide. Both of these changes if carried far enough can result in so lowering the oxygen content or raising the carbon dioxide content that an explosion can not take place even if an explosive proportion of methane is present. Therefore, knowledge of the propagation of flame in limit mixtures of methane, carbon dioxide, oxygen, and nitrogen is important.

Haldane <sup>a</sup> was the first to publish results of experiments bearing on this point. The composition of those mixtures containing the smallest percentages of oxygen that he found would completely inflame follows. Haldane presented his results in terms of black damp, fire damp, and air. In the following table his results have been recalculated to show the percentages of carbon dioxide, oxygen, methane, and nitrogen:

*Inflammable mixtures containing small percentages of oxygen.*

Mixture No.—	CO <sub>2</sub> .	O <sub>2</sub> .	CH <sub>4</sub> .	N <sub>2</sub> .
	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>
1.....	4.21	11.90	7.46	76.43
2.....	3.58	13.23	6.85	76.34
3.....	3.41	13.63	6.46	76.50

Haldane's experiments were made with gas obtained from a coal mine, which was passed into a cylinder, the size of a lamp chimney, from above. Ignition was effected by a small flame from below. Others who have worked on this phase of the subject have been Harger, <sup>b</sup> Leprince-Ringuet, <sup>c</sup> and Burgess and Wheeler, <sup>d</sup> but the

<sup>a</sup> Haldane, J. S., Investigations on the composition, occurrence, and properties of black damp: Trans. Inst. Min. Eng., vol. 8, 1894-95, pp. 549-567.

<sup>b</sup> Harger, John, The prevention of explosions in mines: Trans. Inst. Min. Eng., vol. 43, 1912, pp. 132, 136.

<sup>c</sup> Leprince-Ringuet, F., The inflammability of fire damp and other gases: Coll. Guard., vol. 108, Aug. 14, 1914, p. 376.

<sup>d</sup> Burgess, M. J., and Wheeler, R. V., The propagation of flame in limit mixtures of methane, oxygen, and nitrogen: Jour. Chem. Soc., vols. 105 and 106, November, 1914, pp. 2596-2605.



most exhaustive experiments having to do with the explosibility of methane-air mixtures, in which part of the oxygen of the air was replaced by nitrogen or carbon dioxide or both, have been performed by Clement.<sup>a</sup>

In Clement's first experiments the gases at atmospheric pressure were contained in a Hempel explosion pipette over mercury. The source of ignition, a spark from an induction coil, was applied near the top of the pipette.

Clement found that the limits of inflammability were narrowed as the oxygen was diminished until with 14 per cent of oxygen the low limit was 6.5 per cent of methane and the high limit 6.9 per cent. The inert gas present was nitrogen. When the oxygen was kept constant at 20 per cent and part of the nitrogen replaced by 10 per cent of carbon dioxide the low limit was raised from 5.8 per cent of methane to 6.2 per cent. When the oxygen was again constant at 20 per cent it required the replacement of part of the nitrogen by 62 per cent of carbon dioxide to raise the low limit to 8.8 per cent of methane. Clement found that even when the oxygen was reduced to 17 per cent there was no change in the inflammability of methane-air mixtures from the 5.8 per cent limit observed with 20 per cent of oxygen.

Another set of results was obtained by Clement with a steel tube closed at both ends to hold the mixtures. This tube was provided with stopcocks, an electric-arc igniter, a mixing device, a window for observing the flame, and an opening covered with a paraffin-paper diaphragm through which the pressure was released. The igniter was designed so that an arc could be maintained momentarily or for any desired time. The arc was connected to a 220-volt, direct-current circuit, with suitable resistances in series to give a current of 1.5 amperes, and was placed at the center of the vessel.

With this device the low limit of methane with 19 per cent of oxygen was 5.5 per cent, or 0.3 per cent lower than that found by the first method. With 17 per cent of oxygen the low limit was raised to 5.7 per cent, but even with 13 per cent of oxygen the mixture was explosive with a content of 6.6 per cent as the low limit and 6.8 per cent as the high limit. This second series of experiments again brought out the fact that a large proportion of carbon dioxide was necessary to appreciably affect the limits.

Clement concluded that the action of carbon dioxide in reducing the explosibility of methane-air mixtures can be accounted for by the high specific heat of this gas. He found that carbon dioxide was more effective than nitrogen in reducing explosibility.

The reader should note that atmospheres without enough oxygen (about 17 per cent) to support an oil-fed flame may be explosive if

---

<sup>a</sup> Clement, J. K., The influence of inert gases on inflammable gaseous mixtures: Tech. Paper 43, Bureau of Mines, 1913, 24 pp.

they contain enough methane. For instance, a miner may travel beyond a place where his flame goes out to a place where the methane is present in explosive proportion, and attempt to relight his lamp there. Its flicker just before it became extinguished would have ignited the mixture, were it explosive. Electric sparks in such an atmosphere would be another source of danger.

The limits of inflammability as determined by two of the authors for mixtures of methane and air in which the methane was simply added to the air were 5.5 per cent for the low limit and 14 per cent for the high limit.<sup>a</sup>

In metal mines, gases that make explosive mixtures with air are seldom found except as the result of a mine fire, but dangerous quantities of methane have been found in some metal-mining operations, as in a tunnel investigated by engineers of the Bureau of Mines.<sup>b</sup>

#### CONSUMPTION OF OXYGEN AND PRODUCTION OF CARBON DIOXIDE BY MEN AND BY LIGHTS.

A small proportion of the total carbon dioxide in the air of coal mines is that exhaled by men and given off by lights, and the diminution of oxygen is partly due to consumption of oxygen by these agencies. In a sample of air from an air current of a split where 37 men were working, the authors found by analysis 0.32 per cent CO<sub>2</sub>, 20.47 per cent O<sub>2</sub>, 0.13 per cent CH<sub>4</sub>, and 79.08 per cent N<sub>2</sub>. The quantity of air passing where the sample was collected was 15,000 cubic feet per minute. If one assumes that each man consumed one-seventh of a cubic foot of oxygen per minute and produced one-seventh of a cubic foot of carbon dioxide<sup>c</sup> during the same time, and that each light consumed about 0.03 cubic foot of oxygen per minute and produced the same quantity of carbon dioxide, there would have been consumed by the men and lights about 6.5 cubic feet of oxygen and they would have produced about 6.5 cubic feet of carbon dioxide. These figures represent only 0.04 per cent of the total oxygen consumed and of the total carbon dioxide produced and are practically negligible compared to the oxygen consumed and the carbon dioxide given off by the action of the oxygen of the air on the coal. Of course, if in this part of the mine, more men had been working, or if the quantity of air passing had been less, appreciable vitiation of air would have resulted from the breathing of the men or the burning of the lights.

<sup>a</sup> Burrell, G. A., and Oberfell, G. G., The limits of inflammability of mixtures of methane and air: Tech. Paper 119, Bureau of Mines, 1915, pp. 25-26.

<sup>b</sup> Brunton, D. W., and Davis, J. A., Safety and efficiency in mine tunneling: Bull. 57, Bureau of Mines, 1914, pp. 29-30.

<sup>c</sup> Actually the oxygen absorbed exceeds in small amount the carbon dioxide discharged; the respiratory quotient,  $\frac{\text{CO}_2}{\text{O}_2}$  being equal to about 0.90.

In most metal mines, lights and the breathing of men and of animals are more important in removing oxygen from and adding carbon dioxide to the mine air, because less oxygen is removed and less carbon dioxide is added in other ways.

#### FOULING OF MINE AIR BY DECAY OF TIMBER.

In all mines in which timber is used, some oxygen is taken from the air and some carbon dioxide is added to it by the decay of the timber. This source of black damp is much more important in many metal mines than in most coal mines because of the much larger quantity of timber used, especially in heavy ground. Timber decays through the action of various low forms of plant life, chiefly fungi, some of which take oxygen from the mine air, and practically all of which give off carbon dioxide.

#### SPECIFIC GRAVITY AND COMPOSITION OF BLACK DAMP.

The specific gravity of black damp varies considerably. When methane is present the combined gases may be lighter than air. Great caution should be exercised when any accumulation of black damp lighter than air is found, especially in mines worked with naked lights, as the lesser density is probably due to the presence of methane.

A sample of the following composition was collected by the authors from a cavity in a room of a coal mine:

*Results of analysis of sample of mine air of much less density than atmospheric air.*

CO <sub>2</sub> .....	0.35
O <sub>2</sub> .....	15.82
CH <sub>4</sub> .....	19.32
N <sub>2</sub> .....	64.51
	<hr/> 100.00

Owing to the large amount of methane present, the specific gravity (air = 1) of this sample was only 0.91.

The analysis of another sample showed an accumulation of methane and a deficiency of oxygen in an inclosed section of an anthracite mine that had been sealed for six days because of a fire in an adjoining section. The fire did not affect the particular area from which the sample was obtained because of a heavy intervening roof fall; consequently the sample represented the gases trapped in a stagnant section unaffected by fire. The results of analysis follow:

*Results of analyses of samples of gas in an inclosed area of an anthracite mine.*

CO <sub>2</sub> .....	2.6
O <sub>2</sub> .....	3.0
CH <sub>4</sub> .....	53.0
N <sub>2</sub> .....	41.4
	<hr/> 100.0



The specific gravity of this mixture (air=1) was only 0.77, owing to the large proportion of methane.

Mixtures containing a large amount of carbon dioxide are often found on the floor and in low-lying workings. It happens occasionally that the gas is in a heavy stratum with lighter air above, and these strata are at times so sharply defined that a lighted candle is extinguished at once by lowering it only 1 inch below a certain level.

Equally distinct stratification of gases may be encountered in metal mines. A mixture containing more carbon dioxide, and being consequently heavier than normal air may lie near the floor, or a mixture containing less oxygen and more nitrogen, and being consequently lighter than normal air may accumulate near the roof or in a raise. Either the heavier or the lighter mixture, dangerous to the miner, may be so well stratified that a few inches will make the difference between breathable air and suffocating gases.

Before starting some work near the floor, a miner may hang his lamp well up in the roof, where by burning it indicates that the air near it is breathable; while the air near the floor may be so bad that it would extinguish the lamp immediately. However, except in air directly over a fire area or close to a fire, a large amount of carbon dioxide (over 5 per cent) is unusual in a coal mine. The small proportion of carbon dioxide normally in mine air is indicated by the many analyses of samples presented in this report. Exceptions are samples 5866, 5867, 5785, and 5786 (see p. 40), in which the carbon dioxide percentages were 14.42, 14.36, 19.60, and 19.33, and the specific gravities 1.051, 1.051, 1.071, and 1.069. The samples were collected from sealed mines in which there were or had been fires.

The authors' experience as a result of analyzing gas samples collected from many mines has indicated that if the carbon dioxide results from the action of the oxygen of the air on the coal (including perhaps a small amount released from the pores of the coal), it is not produced in proportions exceeding more than 3 to 5 per cent even after the air has long been in contact with the coal.

The following results of analyses show how small the carbon dioxide content may be in the air of a mine that has been sealed for nine months.

*Results of analyses of gas samples from a mine area that had been sealed for nine months.*

Sample No.	CO <sub>2</sub> .	O <sub>2</sub> .	CH <sub>4</sub> .	N <sub>2</sub> .	Total.	Specific gravity.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
1.....	1.50	0.30	5.29	92.91	100.00	0.95
2.....	1.20	.30	5.37	93.13	100.00	.94

The collectors of the samples wore breathing apparatus. The oxygen had almost entirely disappeared, but only 1.20 and 1.50 per cent of carbon dioxide were present.

#### EFFECT ON LIGHTS OF AIR LOW IN OXYGEN AND HIGH IN CARBON DIOXIDE.

All mine air is somewhat deficient in oxygen as compared with outside air. This deficiency may amount to 0.1 to 2 per cent or more. As the oxygen in air decreases, the illuminating power of lights grows less; the flame of an ordinary oil-fed lamp wick becomes extinguished in air containing about 17 per cent of oxygen.

Haldane <sup>a</sup> determined the effect on the light of a safety lamp when burned in atmospheres deficient in oxygen. The following tabulation prepared by him is of exceptional interest.

*Results of tests to determine diminution of light with diminution of oxygen in surrounding air.*

Light given.	Proportion of oxygen in surrounding air.	Degree of light diminution.	Degree of oxygen diminution.
<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
100	20.93	0	0.00
90	20.66	10	0.27
77	20.34	23	0.59
66	19.88	34	1.05
41	19.34	59	1.89
27	18.92	73	2.01
11	18.28	89	2.65
0	18.01	100	2.92

Haldane observed, in general, that every diminution of 0.1 per cent in the oxygen content of the surrounding air caused a diminution of 3.5 per cent of the value of the light in pure air.

The results of some experiments by the authors with atmospheres that would extinguish a flame are shown below. Lighted candles and acetylene and Wolf lamps were placed in a chamber with air, and when the oxygen content had become so low that the flames were extinguished the residual air was analyzed.

*Results of experiments with atmospheres that would extinguish flame.*

Source of flame extinguished.	Analysis of residual air.		
	CO <sub>2</sub> .	O <sub>2</sub> .	N <sub>2</sub> .
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Candle.....	2.95	16.24	80.81
Acetylene lamp..	6.30	11.70	82.00
Wolf lamp.....	3.00	16.30	80.70

<sup>a</sup> Haldane, J. S., The effects of deficiency of oxygen on the light of a safety lamp: Coll. Guard., vol. 106, Oct. 25, 1912, p. 836.

The carbon dioxide present, which was due to combustion, had no appreciable effect in extinguishing the flame, as was shown by certain experiments performed by the authors. In the experiments varying proportions of carbon dioxide were added to the atmosphere before the flame was introduced. The results of analyses made in connection with the experiments follow:

*Results of experiments to determine effect of carbon dioxide in extinguishing the flame of a candle.*

Experiment No.	Composition of atmosphere at beginning of experiment.		Composition of atmosphere at end of experiment.	
	CO <sub>2</sub> .	O <sub>2</sub> .	CO <sub>2</sub>	O <sub>2</sub> .
1.....	0.04	21.00	2.95	16.34
2.....	3.22	21.13	6.51	16.68
3.....	13.52	20.67	16.00	17.39

The results show that the initial presence of a large amount of carbon dioxide had little effect in extinguishing flame. In all of the laboratory experiments the flame probably lasted a trifle longer than it would have in actual mining practice, because in mines a slight gust of air or a quick movement of a lamp would easily put out a diminishing flame.

For the experiments a certain atmosphere was prepared in a 10-liter bell jar, and the flame was allowed to burn in it until the oxygen had been consumed to a point where the flame would no longer burn. Incidentally there was formed by the combustion a certain amount of carbon dioxide, but the amounts formed had no appreciable effect in extinguishing the flame.

Other experiments to determine the amount of carbon dioxide required to appreciably affect flame extinguishment were made, as follows:

Various mixtures of air and carbon dioxide were prepared in a large 100-liter gas holder, and these mixtures were slowly passed into and out of a small bell jar in which was burning a Wolf miner's safety lamp. The different atmospheres were kept as near constant at 17, 18, 19, 20, and 21 per cent of oxygen as possible, and to each of the mixtures carbon dioxide was added (replacing some nitrogen) until enough carbon dioxide was present to extinguish the flame at the particular oxygen percentage used.

The experiments show simply the effect on flame extinguishment of replacing an equivalent amount of nitrogen with carbon dioxide. Carbon dioxide should have a greater extinguishing power than nitrogen, because of its greater molecular specific heat and its greater conductivity of heat. The percentages of carbon dioxide and of oxygen in the different mixtures that extinguished flame are shown in the following tabulation.



*Percentages of carbon dioxide and of oxygen in different mixtures of the two that extinguished flame.*

CO <sub>2</sub> .	O <sub>2</sub> .	CO <sub>2</sub> .	O <sub>2</sub> .
0.....	16.3	25.....	18.7
5.....	16.9	30.....	19.2
10.....	17.3	35.....	19.6
15.....	17.8	40.....	20.3
20.....	18.3	43.....	21.0

It will be observed that a large percentage of carbon dioxide (replacing the nitrogen) was required to extinguish the flame, 5 per cent of carbon dioxide raising the oxygen content of the extinguishing mixture from 16.3 to 16.9 per cent. This proportion of carbon dioxide in mine air when the oxygen has fallen to 16 or 17 per cent is unusual. It can be said for all practical purposes that the carbon dioxide in mine air has no effect in extinguishing the lamp flames.

Jorrisen <sup>a</sup> has made experiments and compiled data from various sources on flame extinguishment, as shown below:

*Results of Jorrisen's experiments with flame-extinguishing atmospheres.*

Kind of flame.	Proportions in natural extinguishing atmosphere.		Proportions in artificial extinguishing atmosphere.		
	O <sub>2</sub>	N <sub>2</sub>	CO <sub>2</sub> added.	O <sub>2</sub>	N <sub>2</sub>
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Candle.....	16.4	83.6	14	18.1	81.9
Hydrogen.....	6.3	93.7	58	8.8	91.2
Carbon monoxide.....	15.1	84.9	24	16.0	84.0
Methane.....	17.4	82.6	10	18.9	81.1
Ethylene.....	13.2	86.8	26	15.5	84.5
Petroleum.....	16.2	83.8	15	17.9	82.1
Ethyl alcohol.....	16.6	83.4	14	18.1	81.9
Methylated spirits.....	17.2	82.8	13	18.3	81.7
Coal gas.....	11.3	88.7	33	14.1	85.9
Flax.....	16.0	84.0			

The results of experiments of other investigators, as compiled by Jorrisen, follow:

*Results of experiments of various investigators on flame-extinguishing atmospheres.*

Nature of flame.	Investigator.	Residual atmosphere.		
		O <sub>2</sub>	N <sub>2</sub>	CO <sub>2</sub>
		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Candle.....	Clowes.....	15.70	81.10	3.20
Do.....	Burrell.....	16.24	80.81	2.95
Soft candle.....	Harger.....	16.90		
Kerosene.....	Clowes.....	16.60	80.40	3.00
Do.....	Harger.....	16.90		
Petrols spirit.....	do.....	16.90		
Gasoline.....	do.....	15.50		
Alcohol.....	Clowes.....	14.90	80.75	4.35
Methylated spirit.....	do.....	15.60	80.25	4.15
Coal gas.....	do.....	11.35	83.75	4.90
Natural gas.....	Burrell.....	13.90	82.85	3.25
Hydrogen.....	Clowes.....	5.50	94.50	
Do.....	Harger.....	7.50		
Carbon monoxide.....	Clowes.....	13.35	74.40	12.25
Methane.....	do.....	15.60	82.10	2.30
Acetylene.....	Burrell.....	11.70	82.00	6.30

<sup>a</sup> Jorrisen, W. P. The extinction of flames: Jour. Gas Lighting and Water Supply, vol. 130, Apr. 6, 1915, p. 27.

The agreement of most of the results is fairly satisfactory. The experiments are easy to perform, the principal source of error being insufficient mixing of the atmosphere, with its changing  $\text{CO}_2$  and  $\text{O}_2$  content, in which the flame is burning, and consequent nonrepresentative sampling.

Jorrisen calls attention to the fact that there is close similarity between the extinguishment of a flame, such as a burning candle or gas jet, in air as the oxygen decreases and the lessening explosibility of gas mixtures as the oxygen diminishes. If the composition of the air surrounding a burning gas jet is near the extinguishing limit, the temperature of the flame is almost instantly lowered so much that the combustible gas supplied can not be ignited, and the flame goes out. Similarly with explosive mixtures of gas and air in a globe, say, when the upper limit of explosion has been exceeded, the oxygen content has become so small that the flame can not extend itself beyond the source of ignition; that is, the mixture will not explode. This is shown in the following table:

*Data showing relation between oxygen content of upper explosive limits of mixtures of certain gases and air and oxygen content of similar mixtures that will extinguish flame.*

Gas.	Proportion of gas in upper explosive limit. <sup>a</sup>	Proportion of oxygen in gas mixture that extinguishes flame.	Proportion of oxygen at highest explosive limit.
		<i>Per cent.</i>	<i>Per cent.</i>
Hydrogen.....	65	6.50	7.3
Carbon monoxide.....	74	13.4	5.4
Methane.....	14.8	17.4	17.8
Gasoline vapor.....	6.0	15.5	19.6
Natural gas.....	12.0	13.90	18.4
Acetylene.....	75	11.70	5.23
Ethylene.....	22	13.2	16.3

<sup>a</sup> The values in this column were determined by the authors.

The comparison fails badly in the case of carbon monoxide and acetylene.

#### EFFECT OF ATMOSPHERES LOW IN OXYGEN ON FIRES IN MINES.

The effect of the oxygen content of mine air on a fire in a mine is important <sup>a</sup> and is as follows: When a burning section of a mine has been successfully sealed the atmosphere within changes in composition. It first loses oxygen until the content becomes so low, probably about 17 per cent, that flame can not exist. Eventually the oxygen content becomes so small that the rate of combustion slowly decreases, until ultimately the fire is out and the mine can be safely

<sup>a</sup> See Burrell, G. A., and Seibert, F. M., Gas analysis as an aid in fighting mine fires: Tech. Paper 13, Bureau of Mines, 1912, 16 pp.; Burrell, G. A., and Oberfell, G. G., Explosibility of gases from mine fires: Tech. Paper 134, Bureau of Mines, 1916, 31 pp.; Rice, G. S., Mine fires, a preliminary study: Tech. Paper 24, Bureau of Mines, 1912, p. 38.

opened. Conclusions as to the approximate length of time a sealed mine, or part of a mine, should remain closed after a fire has been brought under control are hard to reach, because conditions differ greatly. When the oxygen in an area has been almost or entirely consumed combustion must necessarily stop, and the question then is how long the embers, or partly burned coal, can retain heat sufficient to permit rekindling on the admission of air. Systematic sampling and analysis of the atmosphere in different parts of a sealed mine will show whether the oxygen diminishes regularly, as it should if stoppings are tightly built, or whether air is leaking into the fire area.

#### RESULTS OF ANALYSES OF COAL-MINE ATMOSPHERES.

In the course of the investigations reported in this bulletin a large number of analyses of mine atmospheres were made. Methods of analysis are given in Bulletin 42 of the Bureau of Mines.<sup>a</sup> The analyses were classified in series from 1 to 6, and, for convenience of reference, the original classification has been retained in presenting the results.

The following analysis of pure outside air is presented for comparison with the composition of the mine-air samples analyzed.

##### *Analysis of pure air.*

	Per cent.
Carbon dioxide (CO <sub>2</sub> ).....	0.03
Oxygen (O <sub>2</sub> ).....	20.93
Nitrogen (N <sub>2</sub> ).....	79.04
	<hr/> 100.00

Included in the nitrogen percentage are the so-called rare gases of the atmosphere—argon, helium, neon, krypton, and xenon. These gases constitute about 1 per cent of the total constituents of air and, like nitrogen, are inert.

In the tabulations that follow, the carbon dioxide, oxygen, methane, and nitrogen, as found by analysis, are given. The results of analyses have been recalculated to show the air, black damp, and methane present and the percentage composition of the black damp. The air content is equal to the nitrogen equivalent of the oxygen, according to the proportions found in pure air plus 0.03 per cent of carbon dioxide. The black damp is equal to the total nitrogen minus the nitrogen equivalent of the oxygen plus the carbon dioxide in the mine air in excess of the amount (0.03 per cent) found in pure air.

<sup>a</sup> Burrell, G. A., and Seibert, F. M., The sampling and examination of mine gases and natural gas: Bull. 42, Bureau of Mines, 1913, 116 pp.



The recalculation of the analysis of a mine-air sample will make clear the foregoing statement. The results of analysis of a mine-air sample follow:

*Results of analysis of a sample of air from a coal mine.*

	Per cent.
CO <sub>2</sub> .....	0. 22
O <sub>2</sub> .....	20. 13
CH <sub>4</sub> .....	1. 57
N <sub>2</sub> .....	78. 08
	<hr/> 100. 00

The above results recalculated to show air, black damp, and methane, would give results as follows:

*Results of analysis of mine-air sample recalculated to show air, black damp, and methane.*

Air:	Per cent.	Per cent.
CO <sub>2</sub> .....	0. 03	
O <sub>2</sub> .....	20. 13	
N <sub>2</sub> .....	76. 01	
	<hr/>	96. 17
Black damp:		
CO <sub>2</sub> .....	. 19	
N <sub>2</sub> .....	2. 07	
	<hr/>	2. 26
Methane.....	1. 57	
	<hr/>	1. 57
		<hr/> 100. 00

The composition of the black damp would then be as follows:

CO <sub>2</sub> .....	Per cent..	8. 41
N <sub>2</sub> .....	do.....	91. 59
		<hr/> 100. 00

The air in the above results is found as follows:

$$\text{Nitrogen equivalent of the oxygen} = \left( \frac{79.04}{20.93} \right) 20.13 = 76.01$$

$$\text{Air} = 76.01 + 20.13 + 0.03 = 96.17$$

The black damp in the above results is found as follows: The nitrogen in the sample in excess of that found in pure air = 78.08 - 76.01 = 2.07 per cent. The carbon dioxide in the sample in excess of that found in pure air = 0.22 per cent - 0.03 per cent = 0.19 per cent. The black damp = 0.19 + 2.07 = 2.26 per cent.

RESULTS OF ANALYSES OF SAMPLES IN SERIES 1.

The results of analyses of samples included in series 1 are presented in the following table.

17151°—16—3

TABLE 4.—Results of analyses

Laboratory No.	Source of sample.		Remarks.	Laboratory analysis.			
	Mine No.	Point in mine.		CO <sub>2</sub> .	O <sub>2</sub> .	CH <sub>4</sub> .	N <sub>2</sub> .
3734	1	20 feet from face of No. 8 east entry.	Air still; 1-inch cap on safety lamp.	<i>Per ct.</i> 0.35	<i>Per ct.</i> 19.89	<i>Per ct.</i> 1.94	<i>Per ct.</i> 77.82
3735	1	Face of No. 7 south entry..	Trace of methane shown by safety lamp.	.17	20.34	1.05	78.44
3771	2	Crosscut between 2 rooms.	Quantity of air, 4,200 cubic feet per minute; temperature, 61° F., W. B., 60° F., D. B.; relative humidity, 94 per cent.	.08	20.71	.37	78.84
3773	2	.....do.....	.....do.....	.11	20.64	.23	79.02
3934	3	(a).....	(a).....	7.41	5.76	2.93	83.90
3935	3	(a).....	(a).....	7.51	6.20	2.20	84.09
3981	4	No. 1 room, off No. 10 entry, at roof, 20 feet in by.	Air still.....	1.39	16.79	.07	81.75
3982	4	No. 1 room, off No. 10 entry, at floor, 10 feet in by.	.....do.....	1.17	18.39	.03	80.41
4001	5	Surface crack.....	Crack was over a sealed area in a drift mine that had been on fire a short time previous; fire probably out.	4.82	14.37	.56	80.25
4003	5	Room, 20 feet from a butt, 10 feet from floor.	.....do.....	7.83	10.69	.89	80.59
4071	6	Main intake for booster fan	.....	.35	20.40	.00	79.25
4072	6	Heading of north entry, off 3 east.	.....	.48	20.02	.02	79.48
4073	6	16 east, between rooms 56 and 57.	.....	.53	20.10	.07	79.30
4074	6	Last crosscut between rooms 19 and 20, No. 16 east entry.	.....	1.21	19.39	.07	79.33
4075	6	Main return, at upcast shaft.	.....	1.00	19.31	.05	79.64
4008	7	Sixth left entry at face....	9,900 cubic feet of air per minute.	.09	20.44	1.95	77.52
4009	7	Return air course, near exhaust fan.	23,100 cubic feet of air per minute.	.16	20.63	.42	78.79
3339	8	20 feet inside brattice inclosing former fire area.	{	6.02	6.02	b .16 3.79	} 84.01
3340	8	80 feet inside brattice inclosing former fire area.		7.74	3.18	b .05 3.86	
4138	9	Return air course.....	.....	.19	20.61	.00	79.20
4135	9	.....do.....	.....	.21	20.52	.00	79.27
5607	10	Main return.....	27,000 cubic feet of air per minute; temperature 67° F., W. B. and D. B.; relative humidity, 100 per cent.	.31	20.46	.15	79.08
5609	10	Point in entry where air was considerably diluted with return air from fire area.	.....	3.07	16.40	2.87	77.66
5602	11	Point in a room where it holed through into a manway.	6,900 cubic feet of air per minute; temperature, 76.5° F., W. B., 78° F., D. B.; relative humidity, 92 per cent.	1.53	18.84	2.16	77.47
5603	11	Manway.....	6,900 cubic feet of air per minute; temperature, 74.5° F., W. B., 77.5° F., D. B.; relative humidity, 87 per cent.	.95	19.89	.66	78.50
5604	11	.....do.....	7,200 cubic feet of air per minute; temperature, 74.25° F., W. B., 77.25° F., D. B.; relative humidity, 87 per cent.	1.06	19.73	.80	78.41

<sup>a</sup> Sample was collected at the top of a bore hole through which mine air was issuing. The mine was a drift mine that had been sealed for 30 days because of fire. There were, however, many small openings in the outcrop, making impossible complete exclusion of air.

of coal-mine air samples, series 1.

Recalculated analysis, showing air, black damp, and methane.						Calculated composition of black damp.		Specific gravity of sample (air=1).	
Air.			Black damp.		CH <sub>4</sub>	CO <sub>2</sub> .	N <sub>2</sub> .		
CO <sub>2</sub> .	O <sub>2</sub> .	N <sub>2</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .					
<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	
0.03	19.89	75.11	0.32	2.71	1.94	10.56	89.44	0.99	
.03	20.34	76.81	.14	1.63	1.05	7.91	92.09	.99	
.03	20.71	78.21	.05	.63	.37	7.35	92.65	.99	
.03	20.64	77.94	.08	1.08	.23	6.90	93.10	1.00	
.01	5.76	21.75	7.40	62.15	2.93	10.64	89.36	1.00	
.01	6.20	23.41	7.50	60.68	2.20	11.00	89.00	1.01	
.02	16.79	63.41	1.37	18.34	.07	6.95	93.05	1.00	
.03	18.39	69.45	1.14	10.96	.03	9.42	90.58	1.00	
.02	14.37	54.27	4.80	25.98	.56	15.59	84.41	1.01	
.02	10.69	40.37	7.81	40.22	.89	16.26	83.74	1.02	
.03	20.40	77.04	.32	2.21	.00	12.65	87.35	1.00	
.03	20.02	75.60	.45	3.88	.02	10.39	89.61	1.00	
.03	20.10	75.91	.50	3.39	.07	12.85	87.15	1.00	
.03	19.39	73.22	1.18	6.11	.07	16.19	83.81	1.00	
.03	19.31	72.92	.97	6.72	.05	12.61	87.39	1.00	
.03	20.44	77.19	.06	.33	1.95	15.38	84.62	.99	
.03	20.63	77.91	.13	.88	.42	12.87	87.13	1.00	
.01	6.02	22.73	6.01	61.28	{ b .16 3.79 b .05 3.86 }	8.93	91.07	.99	
.00	3.18	12.01	7.74	73.16		9.57	90.43	1.00	
.03	20.61	77.83	.16	1.37		.00	10.46	89.54	1.00
.03	20.52	77.49	.18	1.78		.00	9.18	90.82	1.00
.03	20.46	77.27	.28	1.81	.15	13.40	86.60	1.00	
.02	16.40	61.93	3.05	15.73	2.87	16.24	83.76	1.00	
.03	18.84	71.15	1.50	6.32	2.16	19.18	80.82	.99	
.03	19.89	75.11	.92	3.39	.66	21.35	78.65	1.00	
.03	19.73	74.51	1.03	3.90	.80	20.89	79.11	1.00	

<sup>b</sup> Laboratory analysis, CO.



TABLE 4.—Results of analyses

Laboratory No.	Source of sample.		Remarks.	Laboratory analysis.			
	Mine No.	Point in mine.		CO <sub>2</sub> .	O <sub>2</sub> .	CH <sub>4</sub> .	N <sub>2</sub> .
5597	12	Main north return.....	15,600 cubic feet of air per minute; temperature, 66° F., W. B., and D. B.; relative humidity, 100 per cent.	<i>Per ct.</i> .34	<i>Per ct.</i> 20.40	<i>Per ct.</i> .23	<i>Per ct.</i> 79.03
5598	12	.....do.....	.....do.....	.31	20.42	.19	79.08
5599	12	Return of 14 north entry..	4,040 cubic feet of air per minute; temperature, 67° F., W. B., and D. B.; relative humidity, 100 per cent.	.12	20.66	.22	79.00
5600	12	.....do.....	.....do.....	.15	20.68	.17	79.00
5578	13	Entry, at face.....	Temperature, 81.75° F.; W. B., 82.75° F., D. B.; relative humidity, 92 per cent.	.35	20.18	1.25	78.22
5579	13	Seventh south entry, 25 feet from face.	6,000 cubic feet of air per minute.	.29	20.25	1.94	77.52
5580	13	Main south return.....	8,398 cubic feet of air per minute; temperature, 78.25° F., W. B., 79.25° F., D. B.; relative humidity, 96 per cent.	.33	20.20	1.42	78.05
5581	13	Last crosscut of fifth north entry.	3,780 cubic feet of air per minute; temperature, 81° F., W. B., 83° F., D. B.; relative humidity, 88 per cent.	.35	20.30	1.08	78.27
5583	13	Return at bottom of air shaft.	45,000 cubic feet of air per minute; 76.50° F., W. B., 77.50° F., D. B.; relative humidity, 96 per cent.	.36	20.37	.78	78.49
5586	13	South side return, 100 feet from air shaft.	5,880 cubic feet of air per minute; 78.25° F., W. B., 78.75° F., D. B.; relative humidity, 98 per cent.	.37	20.21	1.56	77.86
5587	13	Main north return, 100 feet from air shaft.	21,250 cubic feet of air per minute; 76.75° F., W. B., 78° F., D. B.; relative humidity, 99 per cent.	.28	20.41	.67	78.64
5499	14	100 feet outby eighth and ninth right.	Sample represented return air from eighth, ninth, tenth, and eleventh right, off north haulage; 2,000 cubic feet of air per minute; 67° F., W. B.; 68° F., D. B.; relative humidity, 95 per cent.	.08	20.44	.95	78.53
5500	14	.....do.....	.....do.....	.12	20.38	1.03	78.47
5501	14	Between eighth and ninth right in last breakthrough.	Air still.....	.10	20.18	2.04	77.68
5502	14	.....do.....	.....do.....	.16	19.57	4.56	75.71
5503	14	Ninth left, off north haulage, 100 feet from entrance.	Sample of return air from ninth, tenth, eleventh, and twelfth entries, off north haulage; 2,000 cubic feet of air per minute; 67° F., W. B., and D. B.; relative humidity, 100 per cent.	.09	20.39	1.19	78.33
5504	14	.....do.....	.....do.....	.12	20.38	1.03	78.47
5429	15	Third level, close to bulkhead on east side.	Metal mine; 62° F., W. B., 63° F., D. B.; relative humidity, 95 per cent.	.26	20.47	.00	79.27
5430	15	.....do.....	.....do.....	.25	20.44	.00	79.31
5431	15	Seventh level, west, at bulkhead.	.....do.....	.41	20.35	.00	79.24
5432	15	.....do.....	.....do.....	.27	20.53	.00	79.20
5433	15	Upcast shaft, 10 feet below collar.	10,000 cubic feet of air per minute.	.26	20.61	.00	79.13
5434	15	.....do.....	.....do.....	.27	20.60	.00	79.13

of coal-mine air samples, series 1—Continued.

Recalculated analysis, showing air, black damp, and methane.						Calculated composition of black damp.		Specific gravity of sample (air=1).
Air.			Black damp.		CH <sub>4</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .	
CO <sub>2</sub> .	O <sub>2</sub> .	N <sub>2</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .				
<i>Per ct.</i> 0.03	<i>Per ct.</i> 20.40	<i>Per ct.</i> 77.04	<i>Per ct.</i> 0.31	<i>Per ct.</i> 1.99	<i>Per ct.</i> 0.23	<i>Per ct.</i> 13.48	<i>Per ct.</i> 86.52	<i>Per ct.</i> 1.00
.03	20.42	77.11	.28	1.97	.19	12.44	87.56	1.00
.03	20.66	78.02	.09	.98	.22	8.41	91.59	1.00
.03	20.68	78.10	.12	.90	.17	11.76	88.24	1.00
.03	20.18	76.21	.32	2.01	1.25	13.73	86.27	.99
.03	20.25	76.47	.26	1.05	1.94	19.85	80.15	.99
.03	20.20	76.28	.30	1.77	1.42	14.49	85.51	.99
.03	20.30	76.66	.32	1.61	1.08	16.58	83.42	.99
.03	20.37	76.93	.33	1.56	.78	17.46	82.54	.99
.03	20.21	76.32	.34	1.54	1.56	18.09	81.91	.99
.03	20.41	77.08	.25	1.56	.67	13.81	86.19	.99
.03	20.44	77.19	.05	1.34	.95	3.60	96.40	.99
.03	20.38	76.96	.09	1.51	1.03	5.62	94.38	.99
.03	20.18	76.21	.07	1.47	2.04	4.55	95.45	.99
.03	19.57	73.90	.13	1.81	4.56	6.70	93.30	.98
.03	20.39	77.00	.06	1.33	1.19	4.32	95.68	.99
.03	20.38	76.96	.09	1.51	1.03	5.62	94.38	.99
.03	20.47	77.30	.23	1.97	.00	10.45	89.55	1.00
.03	20.44	77.19	.22	2.12	.00	9.40	90.60	1.00
.03	20.35	76.85	.38	2.39	.00	13.72	86.28	1.00
.03	20.53	77.53	.24	1.67	.00	12.57	87.43	1.00
.03	20.61	77.83	.23	1.30	.00	15.03	84.97	1.00
.03	20.60	77.79	.24	1.34	.00	15.19	84.81	1.00

TABLE 4.—Results of analyses

Laboratory No.	Source of sample.		Remarks.	Laboratory analysis.			
	Mine No.	Point in mine.		CO <sub>2</sub> .	O <sub>2</sub> .	CH <sub>4</sub> .	N <sub>2</sub> .
				<i>Per ct.</i> 2.68	<i>Per ct.</i> 13.70	<i>Per ct.</i> .00	<i>Per ct.</i> 83.62
5075	16	20 feet from face and 20 feet east of room neck, 600 feet west of shaft bottom.	Sample taken in area where pillars were being split; air still.				
5076	16	do.	do.	2.66	13.74	.00	83.60
5077	16	Main return air course near shaft.	4,700 cubic feet of air per minute.	1.28	17.70	.00	81.02
5078	16	do.	do.	1.26	17.77	.00	80.97
5486	17	Manway, 6 feet inby.	2,100 cubic feet of air per minute; 76° F., W. B., 79° F., D. B.; relative humidity, 87 per cent.	1.36	18.85	2.08	77.71
5487	17	do.	do.	1.41	18.87	2.06	77.66
5490	17	Manway, 200 feet inby.	2,100 cubic feet of air per minute; 76° F., W. B., 79° F., D. B.; relative humidity, 87 per cent.	1.37	18.84	2.10	77.69
5491	17	do.	do.	1.39	18.86	2.08	77.67
5492	18	North haulage return.	54,720 cubic feet of air per minute; 66.5° F., W. B., 66.5° F., D. B.; relative humidity, 100 per cent.	.08	20.69	.34	78.89
5493	18	do.	do.	.08	20.70	.33	78.89
5465	19	Head of air course.	1,900 cubic feet of air per minute.	1.31	18.42	2.09	78.18
5466	19	do.	do.	.38	20.30	.53	78.79
5467	19	Main return, inby foot of air shaft.	29,500 cubic feet of air per minute; 75.5° F., W. B., 77.75° F., D. B.; relative humidity, 89 per cent.	1.34	18.92	2.06	77.68
5468	19	do.	do.	.42	20.20	.50	78.88
5469	19	15 feet inby F entry.	22,500 cubic feet of air per minute; 75.5° F., W. B., 77.75° F., D. B.; relative humidity 90 per cent.	.31	20.44	.48	78.77
5470	19	do.	do.	1.31	18.40	1.91	78.38
5472	19	do.	do.	1.35	18.66	2.04	77.95
4670	20	No. 1 room, 19th right entry, 4 feet from face and 18 inches from roof.	77° F., W. B. and D. B.; relative humidity 100 per cent.	.10	16.36	5.70	77.84
4671	20	do.	do.	.17	16.45	6.38	77.00
4674	20	Main return, 200 feet from pit mouth.	32,277 cubic feet of air per minute; 53.5° F., W. B. and D. B.; relative humidity 100 per cent.	.24	20.44	.14	79.18
4675	20	do.	do.	.27	20.45	.13	79.15
4556	21	Return air from 8th and 9th right entries.	66° F., W. B. and D. B.; relative humidity 100 per cent.	.08	20.68	.08	79.16
4557	21	do.	do.	.08	20.63	.06	79.23
4558	21	Return air from 8th left entry.	1,000 cubic feet of air per minute; 69° F., W. B. and D. B.; relative humidity 100 per cent.	.09	20.61	.08	79.22
4559	21	do.	do.	.09	20.67	.07	79.17
4509	22	Second crosscut, between rooms 1 and 2, 9th right.	Still air; 65° F., W. B., 66° F., D. B.; relative humidity, 95 per cent.	.21	18.94	6.43	74.42
4510	22	do.	do.	.21	19.07	6.62	74.10
5903	23	Return of one of two splits, 1,500 feet from pit mouth.	40,100 cubic feet of air per minute; 55° F., W. B., 56° F., D. B.	.05	20.85	.00	79.10
5904	23	do.	do.	.08	20.83	.02	79.07
5907	24	Main haulage way, 200 feet from portal.	35,100 cubic feet per minute; 55° F., W. B., 56° F., D. B.; relative humidity, 94 per cent.	.22	20.60	.04	79.14
5908	24	do.	do.	.25	20.57	.01	79.17



*of coal-mine air samples, series 1—Continued.*

Recalculated analysis, showing air, black damp, and methane.						Calculated composition of black damp.			Specific gravity of sample (air=1).
Air.			Black damp.		CH <sub>4</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .		
CO <sub>2</sub> .	O <sub>2</sub> .	N <sub>2</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .					
<i>Per ct.</i> 0.02	<i>Per ct.</i> 13.70	<i>Per ct.</i> 51.74	<i>Per ct.</i> 02.66	<i>Per ct.</i> 31.88					
					<i>Per ct.</i> 0.00	<i>Per ct.</i> 7.70	<i>Per ct.</i> 92.30	<i>Per ct.</i> 1.00	
.02	13.74	51.89	2.64	31.71	.00	7.69	92.31	1.00	
.03	17.70	66.84	1.25	14.18	.00	8.10	91.90	1.00	
.03	17.77	67.11	1.23	13.86	.00	8.15	91.85	1.00	
.03	18.85	71.19	1.33	6.52	2.08	16.94	83.06	.99	
.03	18.87	71.26	1.38	6.40	2.06	17.74	82.26	.99	
.03	18.84	71.15	1.34	6.54	2.10	17.01	82.99	.99	
.03	18.86	71.22	1.36	6.45	2.08	17.41	82.59	.99	
.03	20.69	78.13	.05	.76	.34	6.17	93.83	1.00	
.03	20.70	78.17	.05	.72	.33	6.49	93.51	1.00	
.03	18.42	69.56	1.28	8.62	2.09	17.37	82.63	.99	
.03	20.30	76.66	.35	2.13	.53	13.04	86.96	1.00	
.03	18.92	71.45	1.31	6.23	2.06	12.93	87.07	.99	
.03	20.20	76.28	.39	2.60	.50	14.11	85.89	1.00	
.03	20.44	77.19	.28	1.58	.48	15.05	84.95	1.00	
.03	18.40	69.49	1.28	8.89	1.91	12.59	87.41	.99	
.03	18.66	70.47	1.32	7.48	2.04	15.00	85.00	.99	
.02	16.36	61.78	.08	16.06	5.70	.50	99.50	.97	
.02	16.45	62.12	.15	14.88	6.38	1.00	99.00	.97	
.03	20.44	77.19	.21	1.99	.14	9.55	90.45	.97	
.03	20.45	77.23	.24	1.92	.13	11.11	88.89	1.00	
.03	20.68	78.10	.05	1.06	.08	4.50	95.50	1.00	
.03	20.63	77.91	.05	1.32	.06	3.65	96.35	1.00	
.03	20.61	77.83	.06	1.39	.08	4.14	95.86	1.00	
.03	20.67	78.06	.06	1.11	.07	5.13	94.87	1.00	
.03	18.94	71.53	.18	2.89	6.43	5.86	94.14	.97	
.03	19.07	72.02	.18	2.08	6.62	7.96	92.04	.97	
.03	20.85	78.74	.02	.36	.00	5.26	94.74	.99	
.03	20.83	78.66	.05	.41	.02	10.87	89.13	.99	
.03	20.60	77.79	.19	1.35	.04	12.34	87.66	.997	
.03	20.57	77.68	.22	1.49	.01	12.87	87.13	.997	

TABLE 4.—Results of analyses

Laboratory No.	Source of sample.		Remarks.	Laboratory analysis.			
	Mine No.	Point in mine.		CO <sub>2</sub> .	O <sub>2</sub> .	CH <sub>4</sub> .	N <sub>2</sub> .
				<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>
5866	25	Air-course entry.....	{ Mine sealed when sample was collected because of a previous mine fire.	14.42	3.42	<i>a</i> .41	81.42
5867	25	do.....		14.36	3.63	<i>a</i> .44	81.24
5785	25	{ Fire area in sealed mine, through hole in stopping of outcrop opening.	{	19.60	.90	<i>b</i> 1.95	77.45
5786	25	do.....		19.33	1.18	<i>b</i> 1.99	77.40
5818	26	Main return, 75 feet from mine mouth.	45,024 cubic feet of air per minute; 52° F., W. B., 52.5° F., D. B.; relative humidity, 97 per cent.	.07	20.67	.01	79.25
5819	26	do.....	do.....	.09	20.57	.04	79.30
5822	27	Face of right entry, 1,200 feet west of shaft.	3,600 cubic feet of air per minute; 53° F., W. B., 54° F., D. B.; relative humidity, 94 per cent.	.05	20.53	1.70	77.72
5823	27	do.....	do.....	.08	20.50	1.66	77.76
5730	28	Mouth of 7th butt entry...	9,800 cubic feet of air per minute; 55° F., W. B. and D. B.; relative humidity, 100 per cent.	.12	20.60	.05	79.23
5731	28	do.....	do.....	.13	20.64	.04	79.19
5732	28	Manway, 300 feet south-west of pit mouth.	40,500 cubic feet of air per minute; 52.5° F., W. B., 53° F., D. B.; relative humidity, 97 per cent.	.07	20.74	.05	79.14
5733	28	do.....	do.....	.09	20.69	.05	79.17
5656	29	Main return, 10 feet from bottom of air shaft.	11,300 cubic feet of air per minute; 66° F., W. B., 67° F., D. B.; relative humidity, 95 per cent.	.71	19.85	.40	79.04
5657	29	do.....	do.....	.72	19.87	.43	78.98
5658	29	8½ east entry, at head of last room through from 9 east entry.	1,200 cubic feet of air per minute; 75° F., W. B., 76° F., D. B.; relative humidity, 96 per cent.	1.80	18.16	1.99	78.05
5659	29	do.....	do.....	2.00	17.78	2.21	78.01

*a* CO, 0.33 per cent.*b* CO, 0.10 per cent.

*of coal-mine air samples, series 1—Continued.*

Recalculated analysis, showing air, black damp, and methane.						Calculated composition of black damp.		Specific gravity of sample (air=1).
Air.			Black damp.		CH <sub>4</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .	
CO <sub>2</sub> .	O <sub>2</sub> .	N <sub>2</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .				
<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>
0.00	3.42	12.92	14.42	68.50	c 0.33 .41	17.39	82.61	1.051
.01	3.63	13.71	14.35	67.53	{ c.33 .44	17.53	82.47	1.051
.00	.90	3.40	19.60	74.05	{ c.10 1.95	20.93	79.07	1.071
.00	1.18	4.46	19.33	72.94	{ c.10 1.99	20.95	79.05	1.069
.03	20.67	78.06	.04	1.19	.01	3.25	96.75	.996
.03	20.57	77.68	.06	1.62	.04	3.57	96.43	.996
.03	20.53	77.53	.02	.19	1.70	9.52	90.48	.989
.03	20.50	77.42	.05	.34	1.66	12.82	87.18	.989
.03	20.60	77.79	.09	1.44	.05	5.88	94.12	.996
.03	20.64	77.94	.10	1.25	.04	7.41	92.59	.994
.03	20.74	78.32	.04	.82	.05	4.65	95.35	.996
.03	20.69	78.13	.06	1.04	.05	5.45	94.55	.996
.03	19.85	74.96	.68	4.08	.40	14.29	85.71	.997
.03	19.87	75.04	.69	3.94	.43	14.90	85.10	.997
.03	18.16	68.58	1.77	9.47	1.99	15.75	84.25	.994
					•			
.03	17.78	67.14	1.97	10.87	2.21	15.34	84.66	.994

c Laboratory analysis, CO.



## COMMENT ON ANALYSES OF SAMPLES IN SERIES 1.

The results of analysis of 111 samples of mine air from 29 coal mines are shown in the preceding table. The percentage of black damp in the samples varies from 0.14 to 82.92. The average percentage of carbon dioxide in the black damp is 11.5 per cent, and the average percentage of nitrogen is 89.5 per cent.

The principal factors that probably affect the formation of black damp in coal mines are (1) time of contact between the air and coal; (2) rate of reaction of a particular coal with oxygen, with subsequent formation of some carbon dioxide; (3) temperature of the coal and the air, and (4) presence of moisture. Of these the two first are undoubtedly the most important; hence it is to be expected that different percentages of black damp will be found at different places in the same mine. In a return airway, where the velocity of the air current is high, and the air has been in contact with the coal only a short time, small percentages of black damp are found; however, in the same mine, at a working face, where the air travels slowly, or in an abandoned part of a mine, where the air is stagnant, the percentages of black damp may be high, conditions clearly indicated by many of the results of analyses in the foregoing tables.

If all coals upon reaction with air produced carbon dioxide in the same ratio to the oxygen consumed, then the percentage composition of black damp should be always the same in all mines. Laboratory experiments show, however, that there is no such constancy, as some coals produce more carbon dioxide in relation to the oxygen absorbed than do others; hence black damp formed by the action of air on different kinds of coal must vary in composition.

A question of interest arises, Should the ratio between the carbon dioxide produced and the oxygen consumed be always the same for each coal? If one takes a sample of fresh coal and treats different portions of it with oxygen under the same conditions, practically identical results will be obtained. If, however, one portion is allowed to weather a long time, or if more moisture is present in one than in another, the results are not so concordant. Probably, also, other factors must be taken into consideration. The analyses given in the preceding table show a fair degree of concordance; that is, samples of black damp from different parts of the same mine do not vary as regards percentage composition of the black damp nearly so much as do samples from different mines, although there is lack of strict concordance which shows that even in the same mine different samples differ in composition.

## TEMPERATURE AND HUMIDITY.

Wet and dry bulb temperature readings were taken in 22 of the 29 mines represented in the first series of results. In 6 mines of the 22 the temperature was higher than it should be under the best ventilating conditions. In mine 11 the wet-bulb temperature at one place varied from  $74.25^{\circ}$  to  $78.5^{\circ}$  F. However, this place was used only for traveling. The wet-bulb temperature in mine 13, both at the face and in the moving air current, was too high. The proportion of methane was also higher than it should be, ranging from 0.67 per cent in the return 100 feet from the air shaft to 1.94 per cent 25 feet from the seventh south entry. In other words, the introduction of more fresh air into this mine would not only have cooled it, but would have diluted the methane present.

Mine 16 was poorly ventilated. In the return air course where only 4,700 cubic feet of air was passing (samples 5077 and 5078) the carbon dioxide averaged 1.27 per cent and the oxygen 17.74 per cent. More air would certainly have been desirable.

In mine 18 there were high temperatures combined with high carbon dioxide, low oxygen, and high methane content in the air of the manway.

In mine 20, the oxygen content of the air was low, the carbon dioxide and methane contents high, and the temperature high.

## RESULTS OF ANALYSES OF SAMPLES IN SERIES 2 TO 6.

In Table 5 following are presented the results of analyses representing mine air collected in different mines in this country. The results have been divided into five series following series 1 in the foregoing table. The samples in series 2 and 3 were collected in mines in West Virginia. The samples in series 4 were collected in an anthracite mine in eastern Pennsylvania, and the samples in series 5 and 6 were collected in two different mines in Indiana. In each mine the sampler started at the intake, and traveled in the direction of the air, collecting samples at certain intervals, so as to show the progressive change in the mine air as it traversed the mine and came in contact with more and more coal.

The table shows the laboratory number of each sample; the place where the sample was collected; the composition of the sample as received; the recalculated analysis showing the atmospheric air, black damp, and methane in the sample; and the composition of the black damp. Certain observations made in connection with the different samples are also given.

TABLE 5.—Results of analyses of

SAMPLES IN SERIES 2, FROM

Bottle No.	Sam-ple No.	Distance of point of sam-pling from intake shaft.	Point of sampling.	Quantity of air per minute.	Tempera-ture at point of sampling.		Rela-tive hu-midity.
					Wet bulb.	Dry bulb.	
		<i>Feet.</i>		<i>Cubic feet.</i>	<i>° F.</i>	<i>° F.</i>	<i>Per ct.</i>
1	6358	2,475	Main air course of west heading.....	62,040	54	54	100
2	6357	3,000	Air course, off Green heading.....	16,834	54	54	100
3	6355	3,400	Air course, off Green heading, between rooms 5 and 6.....	20,216	54	55	94
4	6356	3,800	Air course, off Green heading, between rooms 14 and 15.....	13,017	53	54	94
5	6354	4,200	Air course, off Green heading, opposite room 24.....	9,928	53	55	88
6	6361	4,600	Air course, off Green heading, opposite room 33.....	4,260	54	56	88
7	6353	5,000	Air course, off Green heading, opposite room 42.....	Air still.	54	56	88
8	6360	5,575	Ritter heading, opposite room 48.....	5,266	56	57	94
9	6362	6,075	Ritter heading, right side, opposite room 1.....	Air still.	55	56	94
10	6352	6,250	Ritter heading, room 55, off left side, at face.....	Air still.	56	58	89
11	6359	6,275	Ritter heading, at face of room 2, off right side.....	Air still.	55	56	94
12	6350	6,575	Ritter heading, 30 feet from upcast air shaft.....	5,280	55	55	100

SAMPLES IN SERIES 3, FROM

1	6372	1,012	12 left air course.....	Air still.	57	58	94
2	6365	1,512	Face of air course of X entry, off tenth left heading.....	Air still.	58	59	94
3	6366	1,812	Face of room 3, off tenth right heading.....	Air still.	58	60	89
4	6369	2,638	Opposite first cross-cut beyond twelfth left heading.....	Air still.	58	59	94
5	6373	3,438	Between rooms 2 and 3, off twelfth left heading.....	Air still.	57	58	94
6	6367	3,650	Room 3, off twelfth left heading.....	Air still.	57	58	94
7	6368	3,738	Room 7, off twelfth left heading.....	Air still.	58	60	89
8	6374	4,250	Between first and second crosscuts.....	Air still.	57	58	94
9	6375	5,025	Tenth left entry, opposite fifth crosscut.....	1,750	57	58	94
10	6370	5,812	Main air course at intersection with ninth right heading.....	29,000	57	58	94
11	6377	6,612	Main air course, between third and fourth crosscuts, beyond eighth right heading.....	24,450	56	57	94
12	6378	7,425	Main air course, 40 feet beyond seventh left heading.....	41,800	57	58	94
13	6376	8,075	Main air course, 60 feet in by fan.....	61,900	50	54	76

SAMPLES IN SERIES 4, FROM AN ANTHRACITE

1	6293	200	200 feet from slope.....	6,400	61	63.5	87
2	6290	470	470 feet from foot of slope.....	8,800	61	64.5	82
3	6296	682	682 feet from foot of slope, at face.....	Air still.	61	63.5	87
4	6297	802	802 feet from foot of slope.....	12,300	61	63.5	87
5	6298	1,527	1,527 feet from foot of slope.....	10,500	62	64.5	87
6	6301	2,242	2,242 feet from foot of slope.....	9,700	62	64.5	87
7	6289	3,207	3,207 feet from foot of slope.....	8,900	64.5	66	87
8	6302	4,320	3,800 feet from foot of slope.....	6,310	62	64.5	87
9	6292	4,840	4,320 feet from foot of slope.....	4,360	63	66	85
10	6300	5,110	5,110 feet from foot of slope.....	Air still.	63	66	85
11	6299	5,915	5,915 feet from foot of slope.....	7,170	63	66	85
12	6294	6,575	6,575 feet from foot of slope.....	15,300	63	66	85



*mine-air samples, series 2 to 6.*

## A MINE IN WEST VIRGINIA.

Composition of sample as received.				Recalculated analysis to show air, black damp, and methane.						Composition of black damp.		Specific gravity (air=1).
				Air.			Black damp.		Methane.			
CO <sub>2</sub> .	O <sub>2</sub> .	CH <sub>4</sub> .	N <sub>2</sub> .	CO <sub>2</sub> .	O <sub>2</sub> .	N <sub>2</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .	CH <sub>4</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .	
<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	
0.05	20.83	0.03	79.09	0.03	20.83	78.66	0.02	0.43	0.03	4.4	95.6	1.00
.06	20.80	.03	79.11	.03	20.80	78.54	.03	.57	.03	5.0	95.0	1.00
.06	20.85	.04	79.05	.03	20.85	78.73	.03	.32	.04	8.6	91.4	1.00
.07	20.86	.02	79.05	.03	20.86	78.77	.04	.28	.02	12.5	87.5	1.00
.04	20.90	.02	79.04	.03	20.90	78.92	.01	.12	.02	7.7	92.3	1.00
.10	20.76	.00	79.14	.03	20.76	78.39	.07	.75	.00	8.5	91.5	1.00
.11	20.75	.02	79.12	.03	20.75	78.36	.08	.75	.03	9.6	90.4	1.00
.11	20.71	.03	79.15	.03	20.71	78.20	.08	.95	.03	7.8	92.2	1.00
.09	20.81	.00	79.10	.03	20.81	78.58	.06	.52	.00	10.3	89.7	1.00
.29	20.38	.02	79.31	.03	20.38	76.96	.26	2.35	.02	10.0	90.0	1.00
.10	20.77	.05	79.08	.03	20.77	78.43	.07	.65	.05	9.7	90.3	1.00
.15	20.57	.04	79.24	.03	20.57	77.85	.12	1.56	.04	7.1	92.9	1.00

## A MINE IN WEST VIRGINIA.

0.47	19.72	0.00	79.81	0.03	19.72	74.47	.44	5.31	0.03	7.7	92.3	-----
.52	19.64	.02	79.82	.03	19.64	74.17	.49	5.65	.02	8.0	92.0	-----
.42	19.81	.02	79.75	.03	19.81	74.81	.39	4.94	.02	7.3	92.7	-----
.46	19.98	.00	79.56	.03	19.98	75.45	.43	4.11	.00	9.5	90.5	-----
.43	19.82	.00	74.85	.03	19.82	74.85	.40	4.89	.03	7.6	92.4	-----
.45	19.66	.02	79.87	.03	19.66	74.02	.42	5.85	.02	6.7	93.3	-----
.67	19.54	.01	79.78	.03	19.54	73.79	.64	5.99	.01	9.7	90.3	-----
.46	19.63	.08	79.83	.03	19.63	74.13	.43	5.70	.08	7.0	93.0	-----
.51	19.76	.01	79.72	.03	19.76	74.62	.48	5.10	.01	8.6	91.4	-----
.35	20.14	.00	79.51	.03	20.14	76.05	.32	3.45	.01	8.5	91.5	-----
.35	20.24	.02	79.39	.03	20.24	76.43	.32	2.96	.02	9.8	90.2	-----
.40	20.13	.00	79.47	.03	20.13	76.01	.37	3.46	.00	9.7	90.3	-----
.20	20.43	.03	79.34	.03	20.43	77.15	.17	2.19	.03	7.2	92.8	-----

## MINE IN EASTERN PENNSYLVANIA.

0.09	20.80	0.05	79.06	0.03	20.80	78.55	0.06	0.51	0.05	10.5	89.5	-----
.11	20.74	.18	78.97	.03	20.74	78.32	.08	.65	.18	11.0	89.0	-----
.15	20.62	.36	78.87	.03	20.62	77.86	.12	1.01	.36	10.6	89.4	-----
.12	20.68	.19	79.01	.03	20.68	78.09	.09	.92	.19	8.9	91.1	-----
.13	20.69	.24	78.94	.03	20.69	78.13	.10	.81	.24	11.0	89.0	-----
.13	20.65	.34	78.88	.03	20.65	77.98	.10	.90	.34	10.0	90.0	-----
.16	20.48	.65	78.71	.03	20.48	77.34	.13	.65	.65	8.7	91.3	-----
.09	20.52	.80	78.59	.03	20.52	77.49	.06	.80	.80	5.0	95.0	-----
.16	20.49	.94	78.41	.03	20.49	77.37	.13	1.04	.94	11.0	89.0	-----
.19	20.37	1.04	78.40	.03	20.37	76.92	.16	1.48	1.04	9.8	90.2	-----
.20	20.29	1.17	78.34	.03	20.29	76.62	.17	1.72	.24	9.0	91.0	-----
.22	20.39	1.01	78.38	.03	20.39	77.00	.19	1.38	1.01	12.3	87.7	-----

TABLE 5.—Results of analyses of

SAMPLES IN SERIES 5,

Bottle No.	Sample No.	Distance of point of sampling from intake shaft.	Point of sampling.	Quantity of air per minute.	Temperature at point of sampling.		Relative humidity.
					Wet bulb.	Dry bulb.	
		<i>Feet.</i>		<i>Cubic feet.</i>	<i>° F.</i>	<i>° F.</i>	<i>Per ct.</i>
1	6328	50	Main east entry at air course.....	10,100	66	67	95
2	6322	562	Main east entry.....	15,000	66	67	95
3	6321	1,300	do.....	13,680	66	67	95
4	6331	2,060	do.....	7,980	66	67	95
5	6329	2,600	do.....	7,790	65	66	95
6	6330	3,560	No. 1 room, off second west entry.....	Air still.	66.5	67	97
7	6327	4,025	Room 2, off second west entry, off main east entry.....	Air still.	65	66	95
8	6326	4,175	Room 7, second west entry, off main east entry.....	Air still.	66	67	95
9	6323	4,400	Room 9, second west entry, off main east entry, at face.....	Air still.	67	68	98
10	6324	5,100	Second west air course at overcast on first east entry.....	1,470	66	67	95
11	6325	6,500	Return at junction of main east entry and main north entry.....	16,000	67	68	95

SAMPLES IN SERIES 6,

1	6303	(a)	Foot of air shaft, intake air.....	35,700	65	66	95
2	6305	2,000	Fourth west air course.....	9,150	60	61	94
3	6306	3,000	do.....	13,500	61	62	94
4	6307	4,000	do.....	11,425	65	66	95
5	6308	5,300	Face of room in eighth north entry, off fourth west air course.....	Air still.	63	63	100
6	6309	6,300	Face of eighth north entry, off fourth west air course.....	7,200	64	65	95
7	6310	6,900	Room 9, first south entry, off third west air course.....	Air still.	66	67	95
8	6311	7,000	Room 1, at face, off first south entry, off third west air course.....	Air still.	65	66	95
9	6312	8,100	Room 2, near face at breakthrough, off eighth north entry, off fourth west air course.....	8,710	65	66	95
10	6313	9,800	Fourth west air course, return side.....	8,710	65	66	95
11	6314	10,700	do.....	13,000	66	67	95
12	6315	11,600	do.....	15,570	65	66	95
13	6316	13,000	do.....	16,575	65	66	95

a Sample taken at foot of shaft.

mine-air samples, series 2 to 6—Continued.

FROM A MINE IN INDIANA.

Composition of sample as received.				Recalculated analysis to show air, black damp, and methane.						Composition of black damp.		Specific gravity (air=1).
				Air.			Black damp.		Methane.			
CO <sub>2</sub> .	O <sub>2</sub> .	CH <sub>4</sub> .	N <sub>2</sub> .	CO <sub>2</sub> .	O <sub>2</sub> .	N <sub>2</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .	CH <sub>4</sub> .	CO <sub>2</sub> .	N <sub>2</sub> .	
<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	
0.05	20.87	0.00	79.08	0.03	20.87	78.81	0.02	0.27	0.00	6.9	93.1	.....
.08	20.81	.00	79.11	.03	20.81	78.59	.05	.52	.00	8.8	91.2	.....
.06	20.82	.00	79.12	.03	20.82	78.62	.03	.50	.00	5.7	94.3	.....
.09	20.80	.01	79.10	.03	20.80	78.54	.06	.56	.01	9.7	90.3	.....
.19	20.55	.00	79.26	.03	20.55	77.60	.16	1.66	.00	8.8	91.2	.....
.56	19.55	.01	79.88	.03	19.55	73.82	.53	6.06	.00	8.0	92.0	.....
.76	19.04	.07	80.13	.03	19.04	71.90	.73	8.23	.07	8.2	91.8	.....
.74	19.00	.07	80.19	.03	19.00	71.74	.71	8.45	.07	7.8	92.2	.....
.85	18.67	.13	80.35	.03	18.67	70.50	.82	9.85	.13	7.7	92.3	.....
.54	19.53	.04	79.89	.03	19.53	73.75	.51	6.14	.04	7.7	92.3	.....
.55	19.59	.02	79.84	.03	19.59	73.97	.52	5.87	.02	8.1	91.2	.....

FROM A MINE IN INDIANA.

0.04	20.90	0.00	79.06	0.04	20.90	79.06	0.00	0.00	0.00	0.0	0.0	.....
.07	20.78	.10	79.05	.03	20.78	78.47	.04	.58	.10	6.5	93.5	.....
.11	20.70	.10	79.09	.03	20.70	78.17	.08	.92	.10	8.0	92.0	.....
.14	20.67	.13	79.06	.03	20.67	78.05	.11	1.01	.13	9.8	90.2	.....
.25	20.43	.41	78.91	.03	20.43	77.15	.22	1.76	.41	11.1	88.9	.....
.21	20.43	.22	79.14	.03	20.43	77.15	.18	1.99	.22	8.3	91.7	.....
.25	20.31	.26	79.18	.03	20.31	76.69	.22	2.49	.26	8.1	91.9	.....
.49	19.89	.52	79.10	.03	19.89	75.11	.46	3.99	.52	13.9	86.1	.....
.24	20.57	.14	79.05	.03	20.57	77.68	.21	1.37	.14	13.3	86.7	.....
.37	20.36	.29	78.98	.03	20.36	76.88	.34	2.10	.29	13.9	86.1	.....
.76	19.58	.18	79.48	.03	19.58	73.94	.73	5.54	.18	11.6	88.4	.....
.55	19.67	.16	79.62	.03	19.67	74.28	.52	5.34	.16	8.9	91.1	.....
1.27	18.08	.12	80.53	.03	18.08	68.27	1.24	12.26	.12	9.2	90.8	.....



## COMMENT ON RESULTS OF SAMPLING AND ANALYSIS.

## SAMPLES IN SERIES 2.

It will be noted that the first sample of series 2 was collected 2,475 feet from the intake, and the succeeding samples at greater distances, until the last one was taken 6,575 feet from the intake. In general, the first samples were lower in carbon dioxide content and higher in oxygen content, and consequently contained less black damp, but the change was not so marked as in the samples of other series. The composition of the black damp was fairly constant, ranging between 4.4 per cent and 12.5 per cent carbon dioxide and between 87.5 and 95.6 per cent nitrogen. The average composition of the black damp was carbon dioxide, 8.4 per cent; nitrogen, 91.6 per cent.

## SAMPLES IN SERIES 3.

The first eight of the samples in series 3 were collected in still air at working faces. The proportion of carbon dioxide ranged from 0.20 to 0.67 per cent, the oxygen content ranged from 19.54 to 20.43 per cent, and the percentage of black damp ranged from 5.75 to 6.13 per cent, a small but still noticeable increase. The last five samples were collected in the air current and the greater purity of the air is noticeable. The composition of the black damp was remarkably constant, ranging from 6.7 to 9.8 per cent carbon dioxide and from 90.2 to 93 per cent nitrogen. The average composition of the black damp was carbon dioxide, 8.4 per cent; nitrogen, 91.6 per cent.

## SAMPLES IN SERIES 4.

The change in the composition of the air as it traversed the mine is much better indicated by the data regarding the samples in series 4 than by those for the samples in the two preceding series, because the volume of air traversing the place where each sample was collected did not differ enough to influence markedly the percentage change in the carbon dioxide and oxygen contents. Some irregularities due to this cause were, however, noticeable. At the place where sample 6294 was collected, 15,300 cubic feet of air was passing, whereas at the place where sample 6299 was collected, 7,170 cubic feet of air was passing; this difference in the quantity of air passing made the percentage of black damp in sample 6294 (1.57) appear less, whereas in reality the cubic feet of black damp was greater, being 240 cubic feet (15,300 by 1.57) as against 136 cubic feet (7,170 by 1.89) for sample 6299.

## SAMPLES IN SERIES 5.

A noticeable increase in the percentage of black damp is seen in the samples in series 5. Samples 6330, 6327, 6326, and 6323 were collected at working faces in still air, and consequently were much higher in black

damp than the other samples. These samples also showed a progressive increase in black damp content as the distance from the intake was increased. The greatest vitiation in the moving air was noticeable in samples 6324 and 6325.

The composition of the black damp was nearly constant. The average proportion of carbon dioxide was 9.8 per cent, and of nitrogen 90.2 per cent.

#### SAMPLES IN SERIES 6.

Progressive vitiation of the air is also noticeable in the samples of series 6. A noticeable feature is the large increase of black damp in sample 6316, as compared with the proportion in sample 6315. The average percentage of carbon dioxide in the black damp is 10.2 per cent and of nitrogen 89.8 per cent.

#### EFFECTS OF TEMPERATURE AND HUMIDITY AND OTHER FACTORS.

The temperatures in the mine where the samples of series 2 were collected were all moderate, in fact, highly desirable working temperatures. The carbon dioxide content was not high, but the oxygen content was satisfactory and practically no methane was present. No fault could be found with the condition of the air in this mine.

In the mine where the samples of series 3 were collected the carbon dioxide content was higher than in the foregoing mine, the oxygen content lower, and the temperatures a little higher. However, according to modern ideas of the purity of air and the effects of mine air on individuals, one would say the conditions of the air in this mine were all right.

The same is true of the mine where the samples of series 4 were collected, except that the ventilation needed closer watching because of the presence of methane in the air current. A proportion of 1.01 per cent of methane in an airway (sample 6294, where 15,300 cubic feet of air was passing) is rather high. However, the sample represented an anthracite mine in which dust does not constitute the menace that it does in bituminous mines. Many of these mines are extremely gaseous; consequently more difficulty is experienced in keeping the methane down to small percentages than in most bituminous mines.

In the mine where the samples of series 5 were collected conditions were fairly good. At one working face (sample 6323) the oxygen content was 18.67 per cent and the carbon dioxide content 0.85 per cent. The mine was not gaseous. The temperatures were satisfactory.

In the mine where the samples in series 6 were collected, one sample (No. 6316) was high in carbon dioxide (1.27 per cent) and

low in oxygen (18.08 per cent). This sample was collected in the return air where 16,575 cubic feet of air was passing. The temperatures were satisfactory,

#### EFFECTS OF LEAKAGE OF AIR THROUGH DOORS, BRATTICES, ETC.

Necessarily the composition of the air in an entry is influenced greatly by the leakage of air through stoppings, doors, brattices, and overcasts. For instance, the air may be practically normal at a particular place in an entry, not having traveled very far in the mine, and hence not having reacted with much coal. A short distance farther, however, a decided change in its carbon dioxide and oxygen content might be caused by badly vitiated air leaking through a door. That mine will have the tightest doors, overcasts, etc., in which the air shows a gradual and uniform change in composition as it travels through the mine.

#### INFLAMMABLE GAS IN MINES.

Although this report deals primarily with the vitiation of coal-mine air by changes in the content of carbon dioxide and of oxygen, some reference to the presence of inflammable gas (methane) is required, because coal-mine ventilation is largely governed by the quantity of inflammable gas entering the workings.

For instance, the bituminous-mine law of Pennsylvania states that in a nongaseous mine the minimum quantity of air shall be not less than 150 cubic feet per minute for each person employed, and in a mine wherein explosive gas is being generated in such quantities that it can be detected by an approved safety lamp, the minimum quantity of air shall be not less than 200 cubic feet per minute for each person employed therein, and as much more in either case as one or more of the inspectors may deem requisite. Further, under the law mentioned, the use of open lights is prohibited in any entry, airway, traveling way, room, or any other working place where explosive gas is likely to be encountered, and in all such places locked safety lamps must be used exclusively.

In other words, the fact that coal mines generate inflammable gas in greater or less quantities has necessitated the introduction of fresh air to dilute the gas, and this natural gaseous condition of many coal mines has had a great deal to do with keeping the air of coal mines in good condition. The presence of inflammable gas introduces the danger of disastrous explosions, especially in mines where the air is not controlled properly, but this danger, by leading to the continuous introduction of immense volumes of fresh air, year after year, has undoubtedly been the chief element in assuring good ventilation.



In general the air in mines that are termed "gaseous," and consequently are required to have large volumes of air coursing through them and must be inspected more thoroughly to keep all parts of the workings free of inflammable gas, is higher in oxygen, lower in carbon dioxide, lower in temperature, and moves faster, and in all ways has a higher standard of purity than the air in those mines that are termed nongaseous and hence are not required by law to have as much fresh air passing through them, and, in which, because they are not menaced by inflammable gas, the ventilation is not regulated as carefully.

#### **OCCURRENCE AND COMPOSITION OF GAS IN ROCK STRATA OF THE CRIPPLE CREEK GOLD-MINING DISTRICT, COLORADO.**

The senior authors of this paper, while in Colorado on other official business, made a trip to the Cripple Creek gold-mining district to get more data than is at present available regarding the composition of the gas that issues from the rock strata into the gold mines of the district. This gas was said to cause suffocation, and hence to be a menace to life. The data presented herein relate principally to the composition of this gas and its effects on men and on lights.

It is estimated that 25 to 100 miners have been killed by the rock gas in the 25 years that mining has been vigorously carried on at Cripple Creek. Many men have had narrow escapes from death, some of them having been incapacitated for days. In addition, on certain days men can not enter some of the workings. At a few mines fans are used to force outside air into the mines, thus improving conditions in large measure, but even this plan, as used at Cripple Creek, is not entirely adequate at all times.

#### **ORIGIN OF THE STRATA GAS.**

Lingren and Ransome<sup>a</sup> believed that the gas found in the rocks of the Cripple Creek mining district represents the last exhalations of the extinct Cripple Creek volcano. In support of this they stated that little timbering is used in the mines, hence one can not account for the decrease of oxygen or increase of carbon dioxide by oxidation of timber. Further, there is only a small proportion of pyrite and carbonates present in the rocks and ores as compared to that in many other mines entirely free from gas. Moreover, no gas occurs in the oxidized zone, showing that oxidation can not have anything to do with it, and as the gas increases with depth it must be mainly accumulated in strata at depths below those penetrated by the deepest mines of this district.

---

<sup>a</sup> Lingren, W., and Ransome, F. L., *Geology and gold deposits of the Cripple Creek mining district: U. S. Geol. Survey, Prof. Paper 54, 1906, p. 257.*

## EFFECTS OF BAROMETRIC PRESSURE ON OUTFLOW OF STRATA GAS.

The gas is confined in the rock under such low pressure that variations in the outside atmospheric pressure may materially affect the outflow of gas into the mines. Also, at these mines where fans force air in, thereby putting the workings under about 6 or 7 inches of water pressure (about 0.5 inch of mercury), the workings are more or less free from the rock gas which is held back in the pores and crevices of the rocks. The variation in outside atmospheric pressure is easily more than 0.5 inch of mercury, and the effect of the barometric pressure on the outflow of gas is appreciated by mining men.

## COLLECTION OF SAMPLES OF GAS IN FOUR METAL MINES.

One of the authors collected samples of gas in the Mary McKinney, Anaconda, Midget, and Cresson mines. Data regarding the samples are contained in Table 8. General remarks on some of the mines represented in the table are presented as bearing on the data given in the table.

## OBSERVATIONS ON GAS IN MIDGET MINE.

The Midget mine is situated on the west slope of Gold Hill. The workings are partly in breccia and partly in the fine-grained gray gneiss common to the district. The mine has 10 levels. From the shaft to the tenth level is about 900 feet. About 20 men work underground on a leasing system. As regards the men in the levels where samples of gas were collected, four men were in the seventh level, four men on the eighth, and nobody, on account of bad gas conditions, on the ninth or tenth. In the afternoon of the day of sampling the men had to leave the eighth level on account of the gas. Gas conditions were said to be worse than usual at the mine on that day.

The Midget mine uses a pressure system of ventilation. On the surface is a 5-foot Sturtevant fan driven by a 20-horsepower motor which forces air through an air compartment in the shaft to a point below the second level, whence the air spreads into the various levels. In the drifts at various distances from the shaft are air doors, built of 1-inch boards and tightened with canvas. Thus, a small pressure, 3 or 4 inches water gage, is placed on the mine workings, sufficient to check in part the outflow of gas from the rocks and enable the men to do more work than they could without the fan. But when the outflow of gas is stronger than usual, workmen are sometimes driven from the lower levels, and occasionally can not reenter for days.

The Conundrum mine was the first in the Cripple Creek district to install this system of ventilation. This mine adjoins the Midget.

Of the series of samples collected at the Midget mine, only one, No. 664, was collected outside the pressure area or zone and beyond the door. Candles burned freely inside the door, but 20 feet beyond they went out, indicating at this point less than 17 per cent of oxygen. When the door was opened the air rushed through with sufficient strength to blow out the lights.

#### GENERAL OBSERVATIONS REGARDING GASEOUS CONDITIONS IN ANACONDA MINE.

The workings of the Anaconda mine in the town of Anaconda extend through Gold Hill. The mine is worked through an adit having a portal at Anaconda. The prevailing country rock is breccia, with some irregular bodies of latite-phonolite and a few dikes of phonolite and basalt.

The Anaconda shaft is reached by means of an adit 1,135 feet long. The fourth and fifth and lower levels are at times much troubled by rock gas. The progressive vitiation of the air in the fourth drift of the fourth level is shown by the data presented in Table 6 regarding the first six samples from the Anaconda mine. In taking sample 691, 300 feet north of the shaft, which contained 9.02 per cent of oxygen and 5.51 per cent of carbon dioxide, the sampler advanced with an acetylene lamp held near his mouth, until the lamp went out (about 12 or 13 per cent oxygen), then reached down and collected the sample near the floor. The dangerous stratification of the gas is well shown by these analyses. At the height of a man's mouth the air was breathable, but at the floor a man would collapse quickly because of the small amount of oxygen (5.51 per cent) there. Similar stratification is shown by samples taken at other points.

It was noticed on another occasion that a lighted candle would burn fairly well at one point in a drift, but become extinguished further along, although great care was taken that it should not be extinguished by a sudden movement or jerk. At a point still further in the drift it would stay lighted. Usually, though, the nearer one approached the breast of a drift containing much rock gas the worse the air became.

#### OBSERVATIONS ON GAS IN CRESSON MINE.

The Cresson mine is on Raven Hill about two and one-half miles from the Midget mine and two miles from the Anaconda mine. It has 13 levels. The prevailing country rock is andesite breccia.

A wooden door made as tight as possible is placed in each drift beyond the workings and a compressed-air pipe is run through a



hole in the door and is sealed around as tightly as possible. A valve placed inside the door provides for turning on the compressed air. When the air is turned on the rock gas is kept back. The effectiveness of this plan is shown by the analyses of samples collected inside and beyond one of these doors.

In collecting sample 760 the collector advanced 10 feet beyond where the air was comparatively good, holding his breath, then quickly snapped the sealed and evacuated glass sample bottle, whereupon the mine gas immediately filled the bottle, after which he quickly came out. Even so he had a narrow escape, for his knees became weak and his mind slightly hazy, owing no doubt to inadvertently breathing a little of the atmosphere. As soon as he came to better air 10 feet away he felt all right.

Sample 760 showed the effect of the compressed air in holding back the rock gas. At the height of a man's head just at the door the oxygen content of the atmosphere was 16.23 per cent, whereas 10 feet beyond the door the oxygen content was only 2.69 per cent. This sample more closely approximated the pure rock gas than any other collected.

So-called blowers of gas are difficult to find in the Cripple Creek mines. The escape of gas from the rocks usually is uniform from thousands of small pores rather than from large outbursts at one particular place. If one calculates the air-free composition of this sample (No. 760), by eliminating the nitrogen and oxygen according to the proportions found in atmospheric air, the composition becomes 12.69 per cent carbon dioxide and 87.31 per cent nitrogen. These figures show the composition of the pure rock gas, if one assumes that the air in the sample as collected was due to dilution of the rock gas by the mine air.

Apropos of the occasional small outbursts of gas from the rocks, it is interesting to mention at this point an incident related by the superintendent of the Midget mine. In the course of his duties in the mine he was leaning against the wall of a drift with his face close to the rock, talking to another man. Suddenly he felt dizzy and breathless and his knees became weak. Although not knowing the cause of his distress he changed his position and soon felt better. Then it occurred to him that possibly a small feeder of gas was finding its exit close to where he had been standing; so he put his carbide lamp up against the rock at that place. It was immediately extinguished, showing that his assumption had been correct.

#### TABULATED DATA.

Table 6, presenting detailed data regarding the samples taken in the four mines under consideration, follows.

TABLE 6.—Data on gas samples collected in four mines in the Cripple Creek district.

## MIDGET MINE.

Sam- ple No.	Place of taking sample.	Bed.	Temperature.		Relative hu- mid- ity.	Ba- r- ome- ter.	Candle burned.	Acety- lene lamp burned.	CO <sub>2</sub> .	O <sub>2</sub> .	Com- bus- tible gas.	N <sub>2</sub> .	Total.	Remarks.
			Wet bulb.	Dry bulb.										
781	Eighth level, north drift.....	Midget Murray vein.	73	76	P. ct. 88	Inches. 21.6	No.....	Yes...	P. ct. 3.66	P. ct. 16.57	P. ct. 0.00	P. ct. 79.77	P. ct. 100	No men were present at working place.
776	Eighth level, in Olson-Hollengrain slope.						Yes...	Yes...	1.00	18.92	.00	80.08	100	End of air line was 50 feet from slope; top of slope lay at angle of 45° from end of air line.
663	Eighth level, at breast.....	Midget Murray vein.	60	65	75		Yes...	Yes...	.93	20.03	.00	79.04	100	Two men working at breast when sample was taken; compressed air had been turned on this breast for two hours prior to taking sample; breast was 75 feet from end of air line.
770	Raise, 40 feet from breast.....	Intermediate vein.	63.5	69	76		No.....	No a..	8.84	10.06	Trace. <sup>b</sup>	80.30	100	No compressed air; nobody work- ing in this place.
296	Ninth level, north drift, 250 feet from shaft.	Conundrum vein	63	69	76	21.86	Yes c.		2.08	18.53	.00	79.39	100	No men working on ninth level.
769	In shaft at ninth level, 830 feet from surface.		61	66	74				.11	20.79	.00	79.10	100	Air in shaft would not move vane of anemometer.
400	Seventh level, 375 feet northwest of shaft.	Intermediate vein.	63.5	74	59			No d..	8.68	10.86	.00	80.46	100	No men working here.
794	Seventh level, 250 feet north of shaft, 75 feet from breast.	Midget Murray vein.						No...	7.35	11.63	.00	81.02	100	Do.
664	Sixth level, outside of pressure zone, about 200 feet beyond air line.		50.5	54	81				5.09	15.06	.00	79.85	100	

<sup>a</sup> Would burn 1 foot above where sample was collected.<sup>c</sup> Feebly.<sup>d</sup> Burned 2 feet above where sample was taken.<sup>b</sup> Less than 0.02 per cent.

TABLE 6.—Data on gas samples collected in four mines in the Cripple Creek district—Continued.

## ANACONDA MINE.

Sample No.	Place of taking sample.	Bed.	Temperature.		Relative humidity.	Barometer.	Candle burned.	Acetylene lamp burned.	CO <sub>2</sub> .	O <sub>2</sub> .	Combustible gas.		Total.	Remarks.
			Wet bulb.	Dry bulb.							P. ct.	P. ct.		
			° F.	° F.	P. ct.	Inches.		(a)	P. ct.	P. ct.	Trace. <sup>b</sup>	P. ct.	P. ct.	
691	Fourth level, fourth drift, 300 feet north of shaft.	.....	56	59	85	.....	.....	.....	9.02	5.51	Trace. <sup>b</sup>	85.47	100	Nobody working on this level; influx of gas in the afternoons and about one-third of the time in the morning; generally prevents work in this level.
966	Fourth level, fourth drift, 275 feet from shaft.	.....	56	59	85	21.6	.....	.....	7.96	7.50	.02	84.52	100	
937	Fourth level, fourth drift, 250 feet from shaft.	.....	.....	.....	.....	.....	.....	No c.	5.43	11.78	.00	92.79	100	
750	Fourth level, fourth drift, 150 feet from shaft, near floor.	.....	.....	.....	.....	.....	.....	No.	8.09	7.19	Trace. <sup>b</sup>	84.72	100	
394	Fourth level, fourth drift, at roof, 7 feet from floor.	.....	.....	.....	.....	.....	.....	.....	1.64	18.30	.00	80.06	100	
948	Fourth level, fourth drift, 5½ feet from floor.	.....	.....	.....	.....	.....	Yes d.	.....	1.51	18.44	.00	80.05	100	
690	Fifth level, at breast of north drift, 200 feet below fourth level.	.....	64	68	82	.....	No.	Yes...	1.69	17.70	.00	80.61	100	Two men were working here; compressed air had been turned on this breast up to three minutes before sample was taken. Air was being forced into this level from a fan on fourth level.
747	Fifth level, at foot of shaft.	.....	57.5	60.5	85	.....	Yes	.....	.32	20.46	.00	79.22	100	

## MARY MCKINNEY MINE.

Sample No.	Place of taking sample.	Bed.	Temperature.		Relative humidity.	Barometer.	Candle burned.	Acetylene lamp burned.	CO <sub>2</sub> .	O <sub>2</sub> .	Combustible gas.		Total.	Remarks.
			Wet bulb.	Dry bulb.							P. ct.	P. ct.		
			° F.	° F.	P. ct.	Inches.		(a)	P. ct.	P. ct.	Trace. <sup>b</sup>	P. ct.	P. ct.	
761	Eighth level, about 800 feet from surface, No. 15 raise, up about 15 feet from level of drift and about 1,400 feet from shaft.	No. 2 vein. ....	56	60	80	21.87	.....	.....	1.82	10.70	.00	87.48	100	Nobody was working in this raise.
762	Eighth level, 180 feet north of Black vein, No. 12 north drift, 1,500 feet from shaft.	.....	58.5	61	88	.....	.....	No c.	7.54	8.01	.00	84.45	100	Nobody was working at this place; the gas sometimes fills entire drift; conditions were worse than usual at time of sampling.



	Eighth level, at roof, No. 12 north drift, at junction of Black vein	No vein.	58	82	Yes....	4.17	13.87	Trace. <sup>b</sup>	100
790	Junction of No. 2 vein and main crosscut from shaft, 1,250 feet from shaft, <sup>f</sup>	.....	54.5	.....	.....	.25	20.50	.02	.100
797	Eleventh level, at pressure door, 500 feet from shaft, at floor.	.....	69	.....	No....	8.90	7.03	.00	84.07
795	Eleventh level, at pressure door, 5½ feet above floor.	.....	.....	.....	Yes....	2.85	16.23	.00	80.92
760	Beyond air door, about 10 feet beyond samples 797 and 795.	.....	.....	.....	.....	11.08	2.69	.03	86.20
759	Eleventh level, under 1103 slope, 400 feet from shaft.	.....	60	.....	Yes....	2.35	17.19	.00	80.46
771	Eleventh level, at the shaft.....	.....	65.3	.....	.....	.29	20.56	.00	79.15
754	Seventh level, 700 feet from shaft, near bottom of drift.	.....	61	86	No....	1.01	14.05	Trace. <sup>b</sup>	84.94
753	Seventh level, 700 feet from shaft, at top of drift.	.....	61	86	No....	.60	16.74	.00	82.66
772	Seventh level, 700 feet from shaft, at center of drift.	.....	61	86	No....	.62	16.65	.00	82.73
943	Seventh level, 600 feet from shaft.	.....	.....	.....	.....	.39	20.03	.00	79.58
944	Seventh level, 650 feet from shaft.	.....	.....	.....	.....	.51	17.78	.00	81.71

a Carbide lamp went out at roof and floor but burned about  $5\frac{1}{2}$  feet above floor.

*b* Result corresponds with author's laboratory experiments, which showed that acetylene lamp is extinguished when oxygen content of atmosphere drops to about 12 or 13 per cent.

c Less than 0.02 per cent.

*d* Faulty.

*e* Would burn at roof.

*f* Shaft was acting as a downcast, and sample was collected 250 feet from upcast; air seemed good.

## SPECIAL AIR SAMPLES FROM MARY M'KINNEY MINE.

The Mary McKinney mine is situated on the south side of Squaw Gulch, opposite the town of Anaconda. The first ore was shipped in 1893. The workings are mainly in breccia and phonolite. Data on the air samples collected in this mine by one of the authors is shown in Table 8.

At the instance of the authors mine-air samples were collected by A. G. Suydam, a mining engineer of Cripple Creek, for 22 days, almost daily, in the Mary McKinney mine, in the No. 12 north drift, on the 800-foot level, about 1,800 feet from the shaft. The point of sampling was close to a fissure in the rock from which a "feeder" of rock gas intermittently issued.

Table 7 following shows the number of samples collected at this point by Mr. Suydam, date of sampling, results of analyses, and the direction of the wind and the barometric pressure on the date the samples were collected:

TABLE 7.—Data regarding gas samples collected near feeder in Mary McKinney mine.

Sample No.	Date of collection of sample.	Analysis.			Direction of wind.	Barometric pressure.	Condition of weather.	Strength of wind.
		CO <sub>2</sub>	O <sub>2</sub>	N <sub>2</sub>				
	1915.					<i>In. Hg.</i>		
6987.....	Nov. 1	0.24	20.58	79.18	N. to NW.....	22.25	Clear.....	Mild.
6984.....	Nov. 3	12.06	2.97	84.97	SW.....	22.26	.....do.....	Do.
6986.....	Nov. 5	13.27	.94	85.79	S. to SW.....	22.16	Partly cloudy.....	Do.
6981.....	Nov. 9	8.84	7.83	83.33	S. to SW.....	22.06	.....do.....	Do.
6983.....	Nov. 11	.17	20.75	79.08	NW. to N.....	21.96	Clear and cold.....	Strong.
6980.....	Nov. 13	11.05	3.39	85.56	S. to SW.....	21.93	Partly cloudy.....	Mild.
6988.....	Nov. 15	7.86	8.89	83.25	N. to NW.....	22.00	Clear and cold.....	Brisk.
6989.....	Nov. 22	.41	20.64	78.95	N. to NW.....	22.16	Clear.....	Mild.

In addition to the results given above some interesting observations were made by Mr. Suydam as to the effect of different wind and barometer conditions on the air in the drift of the Mary McKinney mine as indicated by candles and acetylene lights. These data are shown in Table 8, following.

TABLE 8.—*Data regarding effect of wind and barometer conditions on air in drift of Mary McKinney mine.*

Date.	Barometer.	Wind.		Sky.	Remarks.
		Direction.	Strength.		
1915.					
Oct. 11	21.875	S. to S. W.	Mild.	Cloudy	Mine closed because of strata gas.
12	21.955	N. to N. E.	Calm.	Clear	Air good in drift.
14	21.985	N. E. to N.	do.	Cloudy	Some strata gas in drift.
15	22.030	N. E. to N.	Strong	do.	Air good.
16	22.130	S. W. to S.	Calm	do.	Do.
17	22.185	N. to N. W.	Mild.	Clear	Do.
19	22.175	do.	do.	do.	Do.
21	22.240	S. to S. E.	do.	do.	Do.
22	22.350	do.	Calm	do.	Some strata gas in drift.
23	22.295	do.	do.	do.	Do.
25	22.200	N. to N. W.	do.	do.	Air good.
26	22.290	N. to N. E.	Light	do.	Do.
27	22.380	N. to N. W.	Calm	do.	Do.
28	22.385	do.	do.	do.	Do.
30	22.235	S. W. to S. to W.	Brisk	Partly cloudy.	Strata gas in drift.
Nov. 2	22.260	W. to S. W. to N. W.	Mild	do.	Air good.
10	21.750	S. to S. W.	do.	Fog	Do.
16	21.800	N. to N. W.	do.	Clear (cold).	Strata gas in drift.
18	do.	do.	do.	do.	No gas; air good.
19	22.185	N. W. to N.	do.	Clear	Air good.

Regarding those samples that were collected and analyzed, the total number of samples collected was too small to permit the drawing of rigid conclusions. With one exception (sample No. 6988) the largest amounts of carbon dioxide and the smallest amounts of oxygen were found in samples collected when the wind was from the south or southwest. This relation agrees with statements of some mining men that when the wind came from the south or southwest the most rock gas entered the mines.

Table 7 shows that rock gas was present in the north drift of the Mary McKinney mine when the wind was south or southwest in 4 cases and when it was north in 2 cases. Also in 10 cases the air was good in the drift when the wind was from the north, and in 3 cases when it was from the south or southwest. Also a consistent relation could not be traced between the barometric pressure and the presence of rock gas.

Some of the mining men of the Cripple Creek district have a theory that when the wind comes from the south or southwest it sweeps up canyons where the rock outcrops. These, being somewhat porous, permit the entrance of atmospheric air, which forces the rock gas into the mine workings.



## COMPOSITION OF STRATA GAS SAMPLES CALCULATED ON AIR-FREE BASIS.

It was impossible to procure samples of pure strata gas as it issued from the rocks in the Cripple Creek mines during the visit of one of the authors. Entrance was made as far as it was possible to penetrate into some of the drifts that were most affected. A sample (No. 760) containing 2.69 per cent oxygen was obtained from the Cresson mine and was the sample containing the largest percentage of strata gas. Undoubtedly if one had been able to penetrate 15 or 20 feet farther into the drift a sample practically devoid of oxygen could have been procured. One can determine rather closely the composition of the strata gas, however, by selecting those samples that contained the smallest percentages of oxygen and calculating them air-free, as has been done in the following table:

*Air-free composition of samples containing small percentages of oxygen.*

## MIDGET MINE.

Sample No.	CO <sub>2</sub> .	O <sub>2</sub> .	Combustible gas.	N <sub>2</sub> .	Total.	Calculated air-free.	
						CO <sub>2</sub> .	N <sub>2</sub> .
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
770	8.84	10.86	0.00	80.30	100.00	18.37	81.63
400	8.68	10.86	.00	80.46	100.00	18.03	81.97
794	7.35	11.63	.00	81.02	100.00	16.53	83.47

## ANACONDA MINE.

966	7.96	7.50	0.00	84.54	100.00	12.40	87.60
937	5.43	11.78	.00	82.79	100.00	12.41	87.59
691	9.02	5.51	.00	85.47	100.00	12.23	87.77
750	8.09	7.19	.00	84.72	100.00	12.32	87.68

## MARY McKINNEY MINE.

762	7.54	8.01	0.00	84.45	100.00	12.20	87.8
976	5.88	10.71	.00	83.41	100.00	12.03	87.97

## CRESSON MINE.

797	8.90	7.03	0.00	84.07	100.00	13.4	86.6
760	11.08	2.69	.00	86.23	100.00	12.71	87.29

These calculations show that the composition of the strata gas varied between 12.03 and 18.37 per cent of carbon dioxide and 81.63 and 87.97 per cent of nitrogen. The average of all of the results is 13.87 per cent of carbon dioxide and 86.13 per cent of nitrogen. Thus, the rock gas is a mixture of carbon dioxide and nitrogen. In making the calculations it was assumed that no oxygen is present

in the gas as it issues from the rock, and that any air in the samples was due to dilution of the rock-strata gas by the air of the mine. This assumption appears to be justified because in a drift more or less filled with rock-strata gas the gas becomes less and less diluted with oxygen as one travels from the good air of the mine farther and farther into the bad air of the drift.

A tendency was noticed among the mining men of Cripple Creek to speak of the strata gas as carbon dioxide or at least as if carbon dioxide was the predominating constituent. This is not the case. Nitrogen is much in excess. The bad effects produced are principally due to the fact that the rock gas so dilutes the air of the mines that the oxygen falls to a point where lights will not burn or so low that life is endangered.

In the authors' opinion the acetylene light should not be used as the sole warning against the presence of gas in these mines. It is true that where the acetylene lamp burns there is enough oxygen (12 to 13 per cent) in the air to support life, but under such conditions the air only a short distance beyond in a drift or at the floor may be fatal to life. The warning of a candle flame affords a much wider margin of safety. At some mines the operators allow no work to be performed where a candle will not burn.

#### EFFECT ON MEN OF PARTIAL PRESSURES OF OXYGEN.

The effects of carbon dioxide and oxygen on men and lights have been discussed in a previous part of this report. A point that can be profitably emphasized here is the effect of the partial pressure of the oxygen on men.

The effects of working in bad air of the Cripple Creek mines are typical of the effects produced by any low oxygen and high carbon dioxide atmospheres. After a day's work the men suffer a feeling of oppression, heaviness, and lassitude, or sleepiness, and a loss of appetite, the degree of distress depending, of course, upon the extent of vitiation of the air. When the air gets very bad, say when a candle will not burn, slight exertion causes breathlessness. Much exposure in bad air brings on headaches and nausea and complete exhaustion. That more fatalities do not occur is due to the fact that the men fairly well appreciate the warning of their lamps and are careful about venturing where acetylene lamps will not burn. Collapse may be very sudden in atmospheres low in oxygen. In fact, it is typical of such atmospheres that little warning is given of their great danger. In some cases men who have collapsed and been rescued have been days recovering. The after effects are very similar to those produced by poisoning by carbon monoxide.

COMPARISON BETWEEN THE INDICATIONS AFFORDED BY CANDLE AND  
BY ACETYLENE FLAME AND ANALYSES OF THE SAMPLES.

At each place where samples were collected by one of the authors he made a note of the condition of his lamp flames, both candle and acetylene. It is interesting to compare these observations with the oxygen percentages of the various samples, as in the following table:

*Data showing relation of oxygen contents of samples to burning of candle and acetylene lamp flames.*

MIDGET MINE.

Sample No.	Oxygen in sample.	Candle burned.	Acetylene lamp burned.	Sample No.	Oxygen in sample.	Candle burned.	Acetylene lamp burned.
	<i>Per cent.</i>				<i>Per cent.</i>		
781.....	16.57	No.....	No.	400.....	10.86	No.....	No.
776.....	18.92	Yes.....	Yes.	794.....	11.63	No.....	No.
770.....	10.86	No.....	No.	664.....	15.06	No.....	Yes.
296.....	18.53	Yes <i>a</i> .....	Yes.				

ANACONDA MINE.

966.....	7.50	No.....	No.	694.....	18.30	Yes <i>a</i> .....	Yes.
937.....	11.78	No.....	No.	948.....	18.44	Yes <i>a</i> .....	Yes.
691.....	5.51	No.....	No.	690.....	17.70	Yes <i>a</i> .....	Yes.
750.....	7.19	No.....	No.				

MARY McKINNEY MINE.

761.....	10.70	No.....	No.	780.....	13.87	No.....	Yes.
762.....	8.01	No.....	No.	976.....	10.71	No.....	No.

CRESSON MINE.

797.....	7.03	No.....	No.	754.....	14.05	No.....	Yes.
795.....	16.23	No.....	Yes.	753.....	16.74	No.....	Yes.
760.....	21.69	No.....	No.	772.....	16.65	No.....	Yes.
759.....	17.19	Yes <i>a</i> .....	Yes.	944.....	17.78	Yes <i>a</i> .....	Yes.

*a* Feebly.

In general these results show that the candle flame became extinguished when the oxygen in the atmosphere fell to 17 to 18 per cent, and that the carbide flame was extinguished when the oxygen content fell to 12 to 14 per cent.

COMBUSTIBLE GAS IN THE ROCK-STRATA GAS.

Eight of the samples examined contained traces of combustible gas. The largest proportion was 0.03 per cent. Presumably all of the samples contained small proportions that could not be detected by analysis. The relation between the contraction and the carbon dioxide content indicated that this combustible gas was methane.



Rarely a small outburst of gas is encountered that burns when a torch is applied to it, but quickly burns itself out.

**VENTILATION CONDITIONS OF 15 METAL MINES AS INDICATED  
BY ANALYSES OF AIR SAMPLES.**

In Table 9 following are shown data regarding samples of air collected in 15 metal mines. Most of the samples were collected by H. M. Wolflin, mining engineer of the Bureau of Mines. Some were collected by Edwin Higgins, also a mining engineer of the Bureau. The samples were analyzed by the authors. The table and the comments following touch upon the ventilation in the different mines.

Calculations to show the amount of black-damp present were made only for samples that were not affected by blasting operations in the mines.

TABLE 9.—Data regarding samples of air collected in certain metal mines.

MINE 1, TONOPAH, NEV.

Lab. No.	Place of sampling.	Temperature.		Relative humidity.	CO <sub>2</sub>	O <sub>2</sub>	N <sub>2</sub>	Black damp.				Composition of black damp.		Remarks.
		Wet-bulb.	Dry-bulb.					CO <sub>2</sub>	N <sub>2</sub>	CO <sub>2</sub> +N <sub>2</sub>		CO <sub>2</sub>	N <sub>2</sub>	
		° F.	° F.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.		P. ct.	P. ct.	
4595	West drift, 1,100-foot level.	73	84	61	.05	20.80	79.15							Blasted 3.15 a. m.; sampled 10 a. m.
4596	Face of east drift, 1,200-foot level.	87.5	90	91	.11	20.65	79.24							Blasted 3 a. m.; sampled 11.20 a. m.; compressed-air ventilation.
4597	21 raise, off crosscut.	87	89	93	.17	20.62	79.21							Blasted 3.15 a. m.; sampled 9.25 a. m.; crosscut is dead end; one man working.
4598	Shaft, above 900-foot level.				.16	20.63	79.21							Taken about 4 p. m., as collector was going up cage.
4599	East drift, No. 7 raise, 1,300-foot level, fan intake.				.07	20.75	79.18							Fan was delivering 1,530 cubic feet of air per minute to within 45 feet of face of drift.
4600	Top of raise, No. 3 east side, 1,400-foot level.	98	99	97	.25	20.68	79.07							Blasted during night; sampled 9.50 a. m.
4601	Shaft, above 900-foot level.				.07	20.70	79.23							Sampled while coming up on cage 11.40 a. m.
4602	Face of crosscut 1055.	78	86	71	.06	20.65	79.29							Blasted 3 a. m.; pipe from fan delivering 510 cubic feet per minute 45 feet back from face; sampled 8.45 a. m.; fan started 7 a. m.
4604	Face of east drift, 400-foot level.	90	95	83										Blasted during night; sampled 10.05 a. m.; temperature of water at sampling place, 101° F.; fan pipe discharging 514 cubic feet of air per minute 20 feet back from face; compressed air also blowing one-half force from ½-inch nozzle. Broken in transit.
4605	Face of east drift, 1,400-foot level (air from fan pipe).	82	92	66	.10	20.58	79.32							Air from fan pipe discharged 20 feet from face.
4606	Face of east drift, shaft vein, off crosscut, 1,200-foot level.	87	89	93	.04	20.84	79.12							Blasted during night; sampled 11.40 a. m.
4609	Face of west drift, off crosscut, 1,200-foot level.	85	91	79	1.37	18.49	80.14							Blasted 3 a. m.; sampled 1.50 p. m., after place had been mucked out; only ventilation from compressed air.
4610	Face of east drift, on 1,300-foot level.	88	95.5	75	.24	20.06	79.70							Sampled 8.15 a. m.; blasted 3 a. m.; men came to work 7 a. m.; end of pipe from fan, 45 feet back from face.
4611	20 feet above sill at top of raise, west drift, shaft vein, off north crosscut, 1,166-foot level.	90	93	89	.31	19.91	79.78							Sampled 10.45 a. m.; blasted 2.30 a. m.; no ventilation but compressed air; one man shoveling.
4613	Intake of fan, raise, east drift, 1,300-foot level.	81	90	69	.12	20.30	79.58							Sampled 8.10 a. m.; blasted during night at 3 a. m.; fan set so as to send the same air to the face repeatedly.

Sampled 8.40 a. m.; blasted between 12 and 3 a. m.; 30 feet ahead of ventilation; only ventilation from 1-inch compressed-air pipe running one-eighth force, 20 feet back from face. Only ventilation from compressed air.

MINE 2, KELLOGG, IDAHO.

4036	Return from slope, near capping station.				1.55	19.44	79.01					Powder smoke rather thick.
4037	do.	81			1.58	19.20	79.22					Considerable smoke, most of it from powder; velocity, 200; area, 6.9 by 6.6; 9,100 cubic feet of air per minute. One man working Waugh stoping drill about 40 feet away; sample lost.
4038	Top floor of slope.	81										Two Waugh stopers had been going 35 minutes; four men working; very dusty.
4039	Top of bench raise.	64			.61	20.10	79.29					Velocity, 655; area, 4.8 by 7.1; air, 22,300; sampled 5 p. m.
4040	Main return, tunnel.	56										Velocity, 643; area, 4.8 by 7.1; air, 21,930; sampled 5.30 p. m.
4041	do.	56			.06	20.84	79.10					Velocity, 655; area, 4.8 by 7.1; air, 22,300; sampled 4.30 p. m.
4042	Return air, tunnel.	56			1.41	19.53	79.06					Velocity, 655; area, 4.8 by 7.1; air, 22,300; sampled 4.30 p. m.
4043	do.	56										Velocity, 655; area, 4.8 by 7.1; air, 22,300; part of air came from another mine.
4044	Crosscut, thirteenth level.											Sampled at beginning of day shift; ventilation entirely dependent on compressed air.
4045	do.	66			.25	20.73	79.03					Sampled 6.25 p. m.; no compressor running; blasted air 4 p. m.
4050	Thirteenth level, face of first drift right, leaving shaft.	66			.23	20.63	79.14					Sampled 7.20 p. m.; no air running; blasted 4 p. m.; air probably turned on and let run half force from 4 to 7 p. m.
4052	Return from slope, near capping station.				1.49	19.29	79.22					Smoke, apparently from fuse.
4053	Intake of No. 2 shaft.	54			.38	20.59	79.03					Velocity, 225; area, 6.5 by 6.2; air, 9,050 c; 30 to 35 men work in this current before it reaches point of sampling.
4054	Breast of raise from twelfth level.	64			.49	20.30	79.21					Small shot fired 12.30 p. m.; sampled 2.20 p. m.; only ventilation from compressed air; not running at time of sampling.
4055	Face of drift on twelfth level, near floor.	66			1.20	19.70	79.10					Sampled 7.10 p. m.; round of holes fired 4 p. m.; air running about one-fourth to one-half force.
4056	Return from slope, near capping station.				1.59	19.30	79.11					Sampled at 5.30 p. m.; not as much smoke as at 5 p. m.; may be going down shaft.

a Velocity of air current in feet per minute,

6 Feet.

c Volume of air current in cubic feet per minute.



TABLE 9.—Data regarding samples of air collected in certain metal mines—Continued.

## MINE 2, KELLOGG, IDAHO—Continued.

Lab. No.	Place of sampling.	Temperature.		Relative humidity.	CO <sub>2</sub>	O <sub>2</sub>	N <sub>2</sub>	Black damp.			Composition of black damp.		Remarks.
		Wet-bulb.	Dry-bulb.					CO <sub>2</sub>	N <sub>2</sub>	CO <sub>2</sub> +N <sub>2</sub>	CO <sub>2</sub>	N <sub>2</sub>	
4077	Top of raise.....	° F. 66		P. ct. ....	P. ct. 0.75	P. ct. 16.43	P. ct. 82.82				P. ct. ....	P. ct. ....	Velocity, 310; area, 3.8 by 5.1; air, 6,000; 60 men working below.....
4078	Slope near bottom of raise.....			84	.30	20.17	79.53	0.27	3.37	3.64	7.4	92.6	Humidity, nearly 100 per cent; dead end, one man working in this place.
4079	.....do.....			81	.13	20.53	79.34	.10	1.82	1.92	5.2	94.8	Humidity, nearly 100 per cent; dead end, one man working in this place.

## MINE 3, BUTTE, MONT.

4099	Return to shaft, south side.....	84	85	96	.29	20.21	79.50	.26	3.19	3.45	7.5	92.5	Velocity, 260; area, 4.40 by 6.5; air, 7,440.
4098	1675 crosscut, return to shaft.....	78.5	80	94	.44	20.02	79.54	.41	3.94	4.35	9.4	90.6	Velocity, 490; area, 3 by 6.1; air, 8,950; sampled, 9.45 a. m.; this is main return to shaft from north side on 1,600 level.
4102	2410 stope, sixth floor.....	83.5	84.5	96	.29	20.26	79.45	.26	2.95	3.21	8.1	91.9	Raise through but no air circulating.
4103	2801 crosscut, 100 feet north of vein A.....	89.5	90.5	96	.19	20.52	79.29	.16	1.81	1.97	8.1	91.9	Only ventilation in this crosscut was from one air drill 700 feet away.
4104	2418-C raise, eleventh floor.....	83	85	92	.31	20.41	79.28	.28	2.21	2.49	11.2	88.8	Sampled, 10 a. m.; blasted, 2 a. m.; air running on tenth floor of raise after blasting; only ventilation, compressed air.
4105	1601 crosscut, at vein near stable.....	75	75.5	98	.06	20.84	79.10	.03	.41	.44	6.8	93.2	Velocity, 165; area, 3.9 by 5.9; air, 3,560; part of main return on 1,600 level.
4106	2402-C drift, near face.....	83	84	96	.18	20.07	79.15	.15	1.10	1.25	12.0	88.0	Only ventilation, compressed air; shot at 2 a. m.; no air turned on till 8.45 a. m., when full head was turned on; sampled, 9.10 a. m., when air felt better near face than back in drift.
4107	Shaft between 400-foot and 800-foot levels.....												Sample taken from cage at 9.30 a. m. before rock-hoisting started.
4108	Return from 2252-C raise, at 2000 level.....	76	76.5	98									Sampled at noon.
4109	Shaft above 700 level, from cage after passing 700-foot level going to surface.....	76	76	100	.10	20.29	79.61						Sampled at 11.15 a. m.; round blasted in face at 2 a. m.; point of sampling 300 to 400 feet ahead of air; two compressed-air hoses blowing air into or near face.
4110	Face of 2601 drift.....	84.5	87	90	.09	20.58	79.33						

MINÉ 4, BUTTE, MONT.

4100	1206 crosscut, 150 feet from 1201 crosscut, at fire wall.	92.5	108.5	60	.42	19.76	79.82	.....	.....	.....	.....	.....	Velocity, 775; area, 1 foot 8 inches by 4 feet 7 inches; air, 6,300; respiration labored where sample was taken.
4101	Third floor of slope 1809.....	79	81.5	90	.35	20.26	79.39	.32	2.89	3.21	10.0	90.0	Floor is 3 sets long and 1 set wide; no raise through to next level.
4096	1200 station of No. 1 shaft.....	84	88	85	.23	20.40	79.37	.20	2.34	2.54	7.9	92.1	Velocity, 1,040; area, 3 feet 10 inches by 6 feet 9 inches; air, 27,200; part of air went through pump room on the 1,200, remainder came up from 1300.
4097	1200 station near junction 1284, return to air shaft.	91.5	96	84.5	.27	20.10	79.63	.24	3.73	3.97	6.0	94.0	Only ventilation, compressed air.
4282	First floor of 1491 slope.....	87.5	88.5	96	.18	20.19	79.63	.15	3.39	3.54	4.2	95.8	Raise was holed at time of sampling; circulation marked.
4283	Twenty-second floor of slope 1637.....	86	87	96	.07	20.85	79.08	.04	.35	.39	10.2	89.8	Ventilation, compressed air only.
4285	Top of fourth floor of 1492 raise.....	85	87.5	92	.19	20.23	79.58	.16	3.19	3.35	4.8	95.2	Two repair men had just left, place when sample was taken.
4286	Sixth floor of slope 1477.....	88	89	96	.15	20.47	79.38	.12	2.08	2.20	5.5	94.5	1414 is air course on 1400; sample taken where air goes into shaft to return to surface.
4224	1414, 10 feet north of slope 1450.....	84	85	96	.11	20.41	79.48	.08	2.41	2.49	3.2	96.8	Velocity, 100; area, 3.8 by 6.7; air, 4,080; air returning to No. 1 shaft.
4225	On 1409 drift at 1453 chute.....	82.5	83.5	96	.15	20.36	79.49	.12	2.61	2.73	4.4	95.6	Two men working in slope near raise which is holed through.
4227	Second floor of slope 1678.....	81	81.25	99	.60	19.73	79.67	.57	5.17	5.74	0.9	90.1	Only ventilation, compressed air, which was running about three-fourths force until two minutes before sample was taken.
4229	1900 level, about 86 feet from No. 1 shaft.	85.5	87	94	.09	20.86	79.05	.06	.28	.34	17.6	82.4	Dead end.
4230	Breast of 2007 drift.....	89.5	90	98	.17	20.55	79.28	.14	1.68	1.82	7.7	92.3	One man working 30 feet from raise which was holed through.
4231	Outlet of main fan, at surface.....	87	89	92	.22	20.41	79.37	.19	2.30	2.49	7.6	92.4	
4232	Eighteenth floor of 1641 slope.....	87	88	.....	.13	20.48	79.39	.10	2.07	2.17	4.6	95.4	

MINE 5, BUTTE, MONT.

4141	Second floor of slope 2390.....	78	80	92	0.07	20.92	79.01	.....	.....	One man running drill (stoper) on third floor near where sample was taken.
4142	Sill floor of slope 2326 (survey point 92).....	85.5	86	99	.14	20.73	79.13	.....	.....	Rather hot; not much air moving, owing to recent blocking of air raise.
4143	Face of 2880 drift, 125 feet west of 2886.....	90	91	96	.17	20.79	79.04	.....	.....	Two 1-inch hose, 30 feet apart, discharging air into place, turned on one-fourth to one-half force; sample taken halfway between them; blasted 2 a. m.; sampled 3 p. m.
4144	Station of 3000 level.....	81	82	96	.16	20.71	79.13	.....	.....	Ventilation by compressed air only; blasted 8 a. m.; sampled 3.40 p. m.
4145	Pan conduit.....	75	75.5	98	.21	20.57	79.22	0.18	1.73	Velocity, 1025; area, 7 by 4.7; air, 33.650.
4146	2017 raise on 1900.....	86	86	100	.35	20.36	79.29	.32	2.41	Velocity, 245; area, 9 by 4.2; air, 9.800.
4147	1900 water crosscut, west of 2017.....	85	88	89	.33	20.42	79.25	.30	2.14	Face is in old work out and filled stoper; compressed air turned on about one-fourth force, 10 feet back from face.
4148	Breast of stope, twelfth floor, 10 sets east of 2270 raise.....	70	72.5	89	.07	20.87	79.06	.04	.25	

TABLE 9.—Data regarding samples of air collected in certain metal mines—Continued.

MINE 5, BUTTE, MONT.—Continued.

Lab. No.	Place of sampling.	Temperature.		Relative humidity.	CO <sub>2</sub>	O <sub>2</sub>	N <sub>2</sub>	Black damp.				Composition of black damp.		Remarks.
		Wet-bulb.	Dry-bulb.					CO <sub>2</sub>	N <sub>2</sub>	CO <sub>2</sub> +N <sub>2</sub>		CO <sub>2</sub>	N <sub>2</sub>	
		° F.	° F.		P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.		P. ct.	P. ct.	
4149	Breast of stope, 2 sets east of 2809 raise.	90	91	96	.13	20.69	79.18	.09	.92	1.01		8.9	91.1	Sampled 2.35 p. m.; blasted on previous shift, 2 a. m.; compressed air running one-third force; no drilling going on.
4152	Face of drift 2825	90	91	96	.13	20.69	79.18	.10	1.05	1.15		8.7	91.3	Compressed air running one-fourth force until 1 minute before sample was taken; pipe delivering 1,720 cubic feet of air per minute 75 feet back from face; place very wet; temperature of water, 100° F.
4153	Raise short distance below fan.	78	78	100	.26	20.36	79.38	.23	2.50	2.73		8.4	91.6	Total air up raise, 41,184 cubic feet per minute.
4154	2444 on 2240 level	82.5	86	87	.17	20.73	79.10	.14	.82	.96		14.6	85.4	Total air up raise, 1,740.
4156	Top of 2370 stope	85	86.5	94	.16	20.72	79.12	.13	.88	1.01		12.9	87.1	Air, 2,518.
4157	Top of 2801 raise on 2400 level	83	85	92	.16	20.65	79.19	.13	1.22	1.35		9.6	90.4	Velocity, 347; area, 3 by 1.5; air, 1,500.
4158	Top of 2835 raise	87.5	89	94	.18	20.64	79.13	.15	1.24	1.39		10.8	89.2	Velocity, 270; area, 2.1 by 2.4; air, 1,300; sampled 10.30 a. m.; considerable leakage of air not included in measurement, so actual quantity was larger than one given.
4159	Top of raise 2209	74	77	87	.08	20.83	79.09	.05	.34	.39		12.9	87.1	Velocity, 373; area, 4.1 by 2.2; air, 3,300.
4161	Top of main air raise on 1100 level	80	80	100	.30	20.35	79.35	.27	2.51	2.78		9.7	90.3	Air, 13,405.
4162	Air raise just below 1000 level	80	80	100	.21	20.54	79.25	.18	1.69	1.87		9.6	90.4	Air, 23,600; area much restricted by lagging placed across the raise by repair men.
4163	1000 level, at door near raise	80.5	81	99	.22	20.55	79.23	.19	1.63	1.82		10.4	89.6	Velocity, 1400; area, 3.9 by 2; air, 23,600; part of this air may have been short circuited from surface.
4164	Incline near surface	78	91	57	.13	20.62	79.25	.10	1.39	1.49		6.7	93.3	Sampled in east hoisting department; shaft downcast in two compartments and upcast in two; no "lining boards" between downcast and upcast air; total up-cast air, 9,320, and total downcast air, 5,070 cubic feet per minute; temperatures at this point in shaft, 79° to 83° F.; sample of upcast air.
4165	1900 water crosscut, west of 2017				.31	20.49	79.20	.28	1.83	2.11		13.3	86.7	Sampled in caved ground, 15 feet in advance of where men were working; timber badly decayed; compressed air running one-half to one-fourth force, about 25 feet back from where sample was taken.



MINE 6, IRONWOOD, MICH.

4315	No. 9 crosscut, 24 level, 225 feet north of rock drift.	60	61	.....	0.14	20.76	79.10	0.11	0.71	0.82	13.4	86.6	Velocity, 450; area, 54; air, 24,300; air passing timbers.
4316	K shaft, 50 feet above 19th level.	55	57.5	.....	.11	20.81	79.08	.08	.50	.58	18.8	86.2	Air, 35,800; air passing timbers.
4317	do.	55	57.5	.....	.06	20.81	79.13	.03	.55	.58	5.2	94.8	Do.
4318	Drift, 347 feet west of 168 raise.	61	62	.....	.10	20.72	79.18	.07	.94	1.01	.....	.....	Velocity, 360; area, 42.25; air, 15,210; air passing timbers.
4319	do.	61	62	.....	.10	20.77	79.13	.07	.70	.77	9.1	90.9	Velocity, 200; area, 42.25; air, 15,210; air passing timbers.
4320	888 feet west of K shaft, 24 level.	59	60	.....	.10	20.84	79.06	.07	.37	.44	15.9	84.1	Velocity, 600; area, 60; air, 36,000; air passing timbers.
4321	do.	59	60	.....	.07	20.78	79.15	.04	.68	.72	5.6	94.4	Do.
4322	Drift, 592 feet west of 168 raise.	61.5	62.5	.....	.09	20.76	79.15	.06	.76	.82	7.3	92.7	Velocity, 265; area, 40.3; air, 10,680; air passing timbers.
4323	do.	61.5	62.5	.....	.10	20.77	79.13	.07	.70	.77	9.1	90.9	Do.
4324	No. 9 crosscut, 24 level, 225 feet north of rock drift.	60	61	.....	.07	20.81	79.12	.04	.54	.58	6.9	93.1	Velocity, 450; area, 54; air, 24,300; air passing timbers.

MINE 7, TROJAN, S. DAK.

4325	Breast of C left stope.	48	49	94	0.26	20.37	79.37	.....	.....	.....	.....	.....	Blasted, 5 a. m.; sampled, 2.25 p. m.
4326	Breast of D stope.	49	51	88	.38	20.17	79.45	.....	.....	.....	.....	.....	Blasted, 5 a. m.; sampled at 2.05 p. m.
4327	Breast of C stope.	46	47	93	.56	19.88	79.56	.....	.....	.....	.....	.....	Fired "plug" at noon; little powder smoke in air at time of sampling, 2.15 p. m.
4328	Raise.	50	51	94	.46	19.95	79.59	.....	.....	.....	.....	.....	Shot a "plug" during morning; sampled, 1.40 p. m.
4329	Breast of M stope.	48	48.5	97	.20	20.31	79.49	.....	.....	.....	.....	.....	Blasted around 5 a. m.; sampled, 1.50 p. m.
4343	West crosscut.	42	43.5	90	.....	.....	.....	.....	.....	.....	.....	.....	Two men in place, one shoveling, one running Leyner drill; blasted in place on previous day shift; sampled, 10.50 a. m.
4344	Tunnel.	42	44	86	.08	20.81	79.11	.....	.....	.....	.....	.....	

MINE 8, TROJAN, S. DAK.

4345	Crosscut raise.	44	46	87	0.12	20.73	79.15	.....	.....	.....	.....	.....	Blasted 18 hours before sample was taken; sampled, 10.20 a. m.
------	-----------------	----	----	----	------	-------	-------	-------	-------	-------	-------	-------	--

MINE 9, TONOPAH, NEV.

4617	Toilet in end of crosscut, south of drift 515, shaft.	.....	.....	.....	0.04	20.88	79.08	.....	.....	.....	.....	.....	
------	---	-------	-------	-------	------	-------	-------	-------	-------	-------	-------	-------	--



Not taken in dead end; considerable air was moving through drift at time of sampling, but it contained some powder smoke from black holing.

Considerable powder smoke from black holing.

Two men working with drill on top of broken rock at 8.30 a. m.

Twelve men working with carbide lamps at point where sample was taken; sample taken just before men returned to work after firing 17 black holes; smoke cleared rapidly.

Boards of moisture hanging on timbers; sampled 11.40 a. m.

Country badly crushed; dead end; odor of decaying timber.

Sample taken in shaft header where powder smoke was thickest.

Crushed country; odor of decaying timber.

Two had been drilling shortly before sample was taken.

MINE 12, VETERAN, NEV.

4433	No. 3 south on 1,250-foot level.....	58	58.5	97	.09	20.69	70.22												
4434	Drift in No. 3 north of vertical slope, 600-foot level.....	78.5	82	80	.34	20.47	70.19	.31	1.90	2.21	11.0	86.0							
4435	1,000-foot level at top raise from No. 6 slope, east 1100.	58	59	95	.13	20.39	70.48												
4436	Eighth floor, No. 9 slope, 1,250-foot level.....	53.5	60.5	95	.19	20.50	70.31	.16	1.91	2.07	7.7	92.3							
4437	Sixth floor, slope 2, 500-foot level.....	59	60	95	.10	20.61	70.29												
4438	Second floor, No. 4 west vertical slope, 500-foot level.....	100.5	101.5	97	1.11	19.24	70.65	1.08	7.00	8.08	13.4	86.6							
4439	Ninth floor, No. 3 north center slope, 600-foot level.....	75	79	84	.10	20.61	70.29												
4441	Shaft header, 600 feet from face, 1,700-foot level.....	61	62	95	.20	20.59	70.21												
4442	Fifth floor, No. 2 north slope, 600-foot level.....	75	79	84	.20	20.43	70.37	.17	2.23	2.40	7.1	92.9							
4443	Fifth floor, No. 2 west slope, 600-foot level.....	76	78	92	.19	20.44	70.37	.16	2.19	2.35	6.8	93.2							
4458	Small raise (No. 3) near breast of No. 59 drift.....	85	88	89	0.47	19.88	70.05	0.44	4.58	5.02	8.8	91.2							
4459	Sublevel 55, 400 feet from face.....	64	65	95	.43	20.11	70.46	.40	3.52	3.92	10.2	89.8							
4460	Bottom 40 winze where drift was being widened.....	64	66.5	89	.61	20.16	70.23	.58	3.11	3.69	15.7	84.3							
4461	Return air at main shaft.....	64	67	86	.69	19.93	70.38	.66	4.12	4.78	13.8	86.2							
4462	Face 34 winze incline.....	73	70	78	1.04	19.37	70.59												
4463	Near bottom of drill hole for suction fan.....	67	68	95	1.10	18.77	80.13	1.07	9.25	10.32	10.1	89.6							
4464	In 85 sublevel just above 59 (No. 3 raise).....				1.22	18.31	80.44												
4465	Face of No. 37 drift (retreating).....	78	81	88	.32	20.38	70.30												
4466	Small raise (No. 3) near breast of 59 drift.....	93	98	83	.71	19.30	70.99	.68	7.11	7.79	8.7	91.3							
4467	Face of sublevel 55.....				.84	19.54	70.02												
4468	Crosscut 4.....		78	92	.57	20.00	70.37	.54	3.62	4.16	13.0	87.0							

Sampled after raise had been connected with sublevel.

Not a dead end; connection had been made for ventilation.

Pairly strong current of air, but velocity not great enough in large area to allow measurement.

Blasted on night shift; 8-inch air pipe connected to fan is delivering 44 cubic feet of air per minute 25 feet back from face.

Velocity, 702; area, 1.9 by 2.15; air, 2,870.

Blasted, 11.25 a. m.

Blasted about midnight; compressed air blowing on eighth floor.

Sample taken 1 inch from top of raise; candle extinguished at top; blasted on night shift; sampled 8.55 a. m.; one man working in raise.

Sampled, 12.35 p. m.; blasted, 11.30 a. m.

Timberman blasted a small "plug" about 15 minutes before sample was taken and turned on compressed air; returned to work and turned off air just before sample was taken.







TABLE 9.—Data regarding samples of air collected in certain metal mines—Continued.

## MINE 14, TONOPAH, NEV.—Continued.

Lab. No.	Place of sampling.	Temperature.		Relative humidity.	CO <sub>2</sub>		O <sub>2</sub>		N <sub>2</sub>	Black damp.				Composition of black damp.		Remarks.
		Wet-bulb.	Dry-bulb.		P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	CO <sub>2</sub>	N <sub>2</sub>	CO <sub>2</sub> +N <sub>2</sub>	CO <sub>2</sub>	N <sub>2</sub>	P. ct.	
4813	No. 24 east drift, 555-foot intermediate level.	° F. 72	° F. 73	91	.30	20.54	79.16	P. ct. 79.05	P. ct. 79.05	.27	1.60	1.87	14.5	85.5	P. ct.	This air is coming from lower levels; sampled 8.50 a. m. Sample of compressed air taken at underground receiver; no compressor or receiver explosions in this mine. Blasted 3.20 p. m.; sampled 10 a. m.; one man drilling, using a Wauight machine requiring about 44 cubic feet of air at pressure of 85 pounds. Sample taken near where timbermen were working inside of where air was blowing.
4814	Top raise, 465-foot level.	69	70	95	.04	20.91	79.02	79.02								
4815	Air receiver on 615-foot level.				.05	20.93	79.02									
4816	20 intermediate stope, 565-foot level.	74	77.5	86	.36	20.39	79.25									
4817	West end of vein near face, crosscut 628, cutting stope.	62	65	86	.14	20.61	79.25			.11	1.43	1.54	7.1	92.9	P. ct.	

## MINE 15, TONOPAH, NEV.

4768	600-C stope.	58	71½	43	0.03	20.36	79.58									Blasted 2.20 a. m.; sampled 8.30 a. m.; stope connected, not a dead end.
4769	501 hangingwall crosscut.	46	58½	41	.07	20.35	79.58									Blasted 3.20 p. m.; sampled 8.40 a. m.; very dusty.
4770	Intake of fan on 600-foot level.				.09	20.81	79.10									No. 23 Sirocco fan so set that some chance for air to be sent to working place more than once; arrangement temporary; sending 1,600 cubic feet of air per minute through fan wheel.
4771	Top raise 662.	76	79½	88	.33	20.36	79.31									Dead end; blasted 3.20 p. m.; sampled 10.36 a. m.; end of ventilating pipe 50 feet down raise and plenty of pipe on the level for miner to bring air to working place; pipe delivered 635 cubic feet of air per minute.
4772	602 drift, about 20 feet ahead of air pipe.	74	78½	84	.10	20.67	79.23	.07	1.18	.07	1.25		3.6	94.4		Dead end; men were working 20 feet ahead of air pipe; delivering 640 cubic feet of air per minute; had not blasted for a day or two.
4773	In main shaft, above the 400-foot level.				.08	20.73	79.19			.05	.91	9.6	3.3	9		Sampled while coming up on cage.
4774	601 winze.	63	72.5	61	.07	20.78	79.14									Blasted 3.20 p. m.; sampled 9 a. m.; dead end, 40 feet down a winze; only ventilation from compressed air.
4775	Raise 548 west.	71	71	100	.71	17.77	81.52	.68	14.42	.68	15.10		4.5	95.5		Candle was extinguished where sample was taken; may have a little gas from strata.





## COMMENTS ON RESULTS OF ANALYSIS.

Only one of the samples (laboratory No. 4609) of air from Mine 1 contained a proportion of oxygen that was either low or excessive as compared with the oxygen content of ordinary air. All of the samples were collected several hours after blasting operations. Hence the mine air was not vitiated in any appreciable degree at the time the samples were collected. All of the temperatures observed at the time of sampling were high—too high to allow men to do good work.

Of the samples collected in Mine 2, four (laboratory Nos. 4036, 4037, 4052, and 4056) contained excessive amounts of carbon dioxide. These samples were all collected at the same place, a few minutes apart. The atmosphere contained powder smoke as the result of a previous blast, hence the atmosphere was vitiated by products of combustion from the blast. Wet-bulb temperatures were not noted in this mine. The dry-bulb temperatures were good except that observed at the time of taking the sample designated laboratory No. 4079. The wet-bulb temperature was not determined in connection with the collection of this latter sample, but the sense of discomfort experienced by the sampler and the moist condition of the place of sampling indicated that the air was almost saturated with moisture. Sample 4079 was the only one, not contaminated with products of combustion from a blast, that had a high percentage of carbon dioxide. The analyses of two of the samples collected in Mine 2 were recalculated to show the percentage of black damp present.

Wet-bulb temperatures observed in Mine 3 exceeded 75° F. The chemical analyses disclosed no low oxygen or high carbon dioxide content. The composition of the black damp ranged from 5.2 to 12 per cent carbon dioxide and from 88 to 94.8 per cent oxygen.

The temperatures and the humidity readings observed in Mine 4 were excessively high. The chemical analyses of the air disclosed no bad condition. Black damp contained 4.2 to 17.6 per cent carbon dioxide and 82.4 to 95.8 per cent oxygen.

The temperatures were higher in Mine 5 than is compatible with the best working efficiency. The chemical analyses disclosed no high carbon dioxide or low oxygen percentages. The black damp contained 6.7 to 13.8 per cent carbon dioxide and 86.2 to 93.3 per cent nitrogen.

Favorable temperatures were found in Mines 6, 7, 8, and 9. The chemical analyses of the air also showed good conditions.

Temperatures in Mine 10 were not so good.

Most of the observed temperatures in Mine 11 were good. There was one noteworthy exception, that noted at the time of taking sample 4438. The carbon dioxide content was also high in that sample.

Considerable powder smoke was present at some places where samples were collected, but no carbon monoxide that might have remained in the air after the shooting was found.

Many of the temperatures observed in Mine 12 were low. Some exceeded 75° F., wet bulb. The carbon dioxide and oxygen contents in some of the samples were low, especially in samples 4463, 4464, and 4473. Just how much blasting operations affected these samples is problematical. One sample (No. 4464), collected 11 hours after blasting, contained 1.22 per cent carbon dioxide and 18.34 per cent oxygen. In general the chemical analyses of the samples from Mine 12 disclosed higher carbon dioxide and lower oxygen contents than in the samples from any of the other mines.

The temperatures in Mine 12 were favorable.

The temperatures observed in Mines 14 and 15 were good. Only in Mine 12 were samples collected that contained excessively high percentages of carbon dioxide or excessively low percentages of oxygen.

In summing up conditions in these metal mines it can be said that, as regards the chemical analyses of the samples collected, the air, on the whole, was very good. Samples collected several hours after blasting, and when men were mucking out their places, did not contain carbon monoxide from the blasting operations. Changes in the air of metal mines due to oxidation are not as rapid as in coal mines, in spite of the fact that immense volumes of air sweep through coal mines as compared to metal mines, where the ventilation is natural or where air is supplied from compressed-air lines.

The chief trouble lies in the high temperatures, both wet-bulb and dry-bulb; also, the stagnation of the air encountered in many places is so bad as to be detrimental to the health and to the efficiency of the workmen.

#### OBSERVATIONS REGARDING BLACK DAMP IN CERTAIN METAL MINES.

The average percentage of black damp in certain mines and the average composition of the black damp are shown in the following table:

*Average percentage and average composition of black damp in certain metal mines.*

Mine No.	Average per cent of black damp.	Composition of black damp.	
		CO <sub>2</sub> .	N <sub>2</sub> .
		<i>Per cent.</i>	<i>Per cent.</i>
2 and 3...	2.52	8.4	91.6
4.....	2.64	7.4	92.6
5.....	1.60	10.9	89.1
6.....	.71	7.9	92.1
11.....	2.78	9.5	90.5
12.....	5.76	11.8	88.2



The average of all analyses in the above table is 2.67 per cent black damp with a composition corresponding to 9.3 per cent carbon dioxide and 90.7 per cent nitrogen.

The average composition of black damp in metal mines is about the same as that found in coal mines. A difference in its occurrence in the two kinds of mines lies in the amount that can be produced in a given time.

If coal mines depended only on natural ventilation or on compressed air from pipes at working places, the amount of black damp produced would usually exceed far more than it does that produced in metal mines, for coal reacts with oxygen much more rapidly than do most of the rocks, wood, etc., found in metal mines.

#### SUMMARY.

The most satisfactory definition of the term "black damp" is an accumulation of carbon dioxide and nitrogen in excess of the percentage found in pure atmospheric air.

The principal factors that affect the changes in mine air are (1) the velocity with which the mine air traverses the mine passages; (2) the amount of coal with which it comes in contact; (3) the gaseous (methane) nature of the seam; (4) the nature of the coal as regards its power to react with oxygen; (5) the temperature and the wetness of the mine.

Carbon dioxide must be present in large proportions before it threatens life. A proportion of 3 to 4 per cent of carbon dioxide in air affects the breathing of most people. Men may, however, work for a long time in such an atmosphere, although their efficiency as workmen will be greatly affected and they will become fatigued quickly. The presence in air of as little as 1 or 2 per cent of carbon dioxide is not so much a matter of safety and comfort to those who breathe it as it is of their efficiency as workmen.

Distress is caused in some people when the oxygen content falls to less than 13 per cent. Rapid breathing is produced much more quickly by an excess of carbon dioxide than by a corresponding deficiency of oxygen. The important point to remember is that rapid breathing caused by carbon dioxide starts long before there is any serious danger, whereas rapid breathing caused by a deficiency of oxygen is a grave symptom and points urgently to serious danger.

By acclimatization people live the year round at high altitudes where the air has an oxygen content, by weight, that is the same as that of an atmosphere at sea level containing 12 per cent oxygen by volume. People unaccustomed to such atmospheres, if suddenly plunged into them, experience severe distress. In an experiment conducted by the authors a man lost consciousness temporarily when the oxygen content of an atmosphere he breathed fell to 7 per cent.

Mice and canaries are about as resistive to low-oxygen atmospheres as men; hence they can not be used by exploring parties to give warning of atmospheres that are dangerously low in oxygen.

An excess of oxygen or a diminution of oxygen such as often occurs in many mines, if present in buildings above ground where people congregate, would be indicative of extremely bad ventilation. However, in mines having such an excess or diminution the ventilation may be excellent. In most buildings above ground, the problem of good ventilation is not to correct a diminished content of oxygen or an increased content of carbon dioxide, but to maintain proper temperature and relative humidity, and to keep the air moving, although the presence of an excessive proportion of carbon dioxide, more than 0.1 or 0.2 per cent, may be a reliable indication of air that will produce injurious effects on men. In these buildings the carbon dioxide comes chiefly from the air exhaled by the persons present, and, if fresh air is not admitted, invariably accompanies bad conditions such as stagnant and oppressive air, high temperature, and frequently high humidity. In coal mines the carbon dioxide is principally from the action of the oxygen of the air on the coal, and 0.2 per cent is frequently found in the cool, swiftly moving air of returns where 50,000 or more cubic feet of air is passing per minute. Hence, a proportion that accompanies good conditions of ventilation in a coal mine may indicate extremely poor conditions of ventilation in the room of a house.

A similar statement applies to oxygen. The oxygen content is scarcely ever normal in a coal mine, owing to the gas being absorbed by the coal; in fact, a diminution of 1 per cent is not uncommon. Such a diminution in a building filled with people would be accompanied by intolerable conditions of ventilation. However, even in coal mines the oxygen content of the air should not be allowed to become too low, and the authors believe that it should not fall below 19 per cent. That this limit can easily be maintained in coal mines is indicated by many mine-air analyses made by the authors. The maximum percentage of carbon dioxide allowed in English coal mines is 1.25 per cent.

The principal cause of the depletion of oxygen in coal-mine air and the increase of carbon dioxide is the reaction between the oxygen of the air and coal. Some of the oxygen is actually held dissolved in the coal substance. Part of the oxygen is converted into water, part into carbon dioxide, and part (by far the larger part) is retained as combined oxygen to give compounds richer in oxygen than the coal itself. Part of the carbon dioxide is retained by the coal.

Explosive proportions of methane in air become nonexplosive when the proportion of oxygen in the atmosphere falls below about 14 per cent. Carbon dioxide has only a slightly greater effect in reducing

the explosibility of methane-air mixtures than nitrogen has; for instance, when the oxygen is kept constant at 20 per cent, part of the nitrogen must be replaced by 10 per cent of carbon dioxide to raise the low limit for methane from 5.8 to 6.2 per cent.

The specific gravity of black damp varies considerably in certain mixtures. When methane is present the combined gases may be lighter than air. Great caution should be observed when one detects an accumulation of black damp of lighter density than air, especially in coal mines in which naked lights are used, as this lesser density is probably due to the presence of methane.

Except directly over a fire area or close to a mine fire, a large proportion of carbon dioxide (more than 3 to 5 per cent) is unusual in the air of a coal mine.

An oil-fed flame becomes extinguished when the oxygen in air falls to about 17 per cent; an acetylene flame is extinguished when the oxygen falls to about 12 or 13 per cent.

Lack of oxygen is the important factor in extinguishing lights. In some experiments conducted by the authors, the oxygen content fell to 16.3 per cent before the flame became extinguished, but the presence of 10 per cent of carbon dioxide raised the extinguishing percentage of oxygen to 17.3.

Atmospheres that do not contain enough oxygen to support an oil-fed flame (about 17 per cent) may be explosive when the oxygen content is as low as 14 per cent, if enough methane is present.

When a burning part of a mine has been successfully sealed the composition of the atmosphere within changes. The oxygen decreases to a proportion (probably about 17 per cent) that will not support flame; ultimately the oxygen content becomes so small that the rate of combustion is extremely low, so low that combustion entirely ceases, the embers cool, and the admission of air when the mine is reopened does not rekindle them.

In 111 samples of gas from 29 mines represented, the average percentage of carbon dioxide in the black damp was 11.5 per cent, and the average percentage of nitrogen 89.5 per cent.

In 6 mines of 22 examined the temperature was higher than it should be (75° F., wet bulb) under the best ventilating conditions.

Analyses of a large number of samples, show how mine air changes as it traverses the workings. The average composition of the black damp was 9.2 per cent carbon dioxide and 90.8 per cent nitrogen. Except for two or three samples, in which carbon dioxide was high and the oxygen low, the quality of the air was good.

As regards the unfavorable effect of black damp on men, on lights, and on the explosibility of methane-air mixtures, the diminution of oxygen in the atmosphere, resulting in the formation of more nitrogen, is mainly responsible. The presence of carbon dioxide is far less



important; hence the objection to making the terms "black damp" and "carbon dioxide" synonymous.

Many of the mines of the Cripple Creek region are menaced with gas that is loosely held in the rock strata and that issues at times into the mines, so that workmen can not enter certain drifts and occasionally a whole mine.

This gas, according to Lingren and Ransome, is of deep-seated origin and probably represents the last exhalations of the extinct Cripple Creek volcano. It contains about 14 per cent carbon dioxide and 86 per cent nitrogen. The gas is confined in the rock strata under very low pressure, so that changes in outside atmospheric pressures affect its outflow.

A limited number of observations made by the authors indicate that the direction of the wind influences the outflow of the gas.

Data regarding ventilation conditions in 15 metal mines other than the Cripple Creek mines are shown. The average percentage of black damp found in these mines was 2.67 per cent. The average composition of the black damp was 9.3 per cent carbon dioxide and 90.7 per cent nitrogen, or about the same as that found in coal mines. In general, the composition of the air in these metal mines, as shown by the carbon dioxide and oxygen content, were good. Difficulty is experienced, however, in keeping down the wet and the dry bulb temperature. In many cases the temperatures were so high as to be detrimental to health.

## PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF COAL MINING.

Limited editions of the following Bureau of Mines publications are temporarily available for free distribution. Requests for all publications can not be granted, and applicants should select only those publications that are of especial interest to them. All requests for publications should be addressed to the Director, Bureau of Mines, Washington, D. C.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls., 28 figs.

BULLETIN 42. The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs.

BULLETIN 45. Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 46. An investigation of explosion-proof mine motors, by H. H. Clark. 1912. 44 pp., 6 pls., 14 figs.

BULLETIN 48. The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell. 1913. 50 pp., 3 pls., 7 figs.

BULLETIN 50. A laboratory study of the inflammability of coal dust, by J. C. W. Frazer, E. J. Hoffman, and L. A. Scholl, jr. 1913. 60 pp., 95 figs.

BULLETIN 52. Ignition of mine gases by the filaments of incandescent electric lamps, by H. H. Clark and L. C. Ilsley. 1913. 31 pp., 6 pls., 2 figs.

BULLETIN 56. First series of coal-dust explosion tests in the experimental mine, by G. S. Rice, L. M. Jones, J. K. Clement, and W. L. Egy. 1913. 115 pp., 12 pls., 28 figs.

BULLETIN 60. Hydraulic mine filling; its use in the Pennsylvania anthracite fields; a preliminary report, by Charles Enzian. 1913. 77 pp., 3 pls., 12 figs.

BULLETIN 62. National mine-rescue and first-aid conference, Pittsburgh, Pa., September 23-26, 1912, by H. M. Wilson. 1913. 74 pp.

BULLETIN 68. Electric switches for use in gaseous mines, by H. H. Clark and R. W. Crocker. 1913. 40 pp., 6 pls.

BULLETIN 69. Coal-mine accidents in the United States and foreign countries, compiled by F. W. Horton. 1913. 102 pp., 3 pls., 40 figs.

BULLETIN 99. Mine-ventilation stoppings, with especial reference to coal mines in Illinois, by R. Y. Williams. 1915. 30 pp., 4 pls., 4 figs.

TECHNICAL PAPER 6. The rate of burning of fuse as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigations of fuse and miners' squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

TECHNICAL PAPER 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp., 1 fig.

TECHNICAL PAPER 14. Apparatus for gas-analysis laboratories at coal mines, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 7 figs.

TECHNICAL PAPER 17. The effect of stemming on the efficiency of explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp., 11 figs.

TECHNICAL PAPER 18. Magazines and thaw houses for explosives, by Clarence Hall and S. P. Howell. 1912. 34 pp., 1 pl., 5 figs.

TECHNICAL PAPER 19. The factor of safety in mine electrical installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 21. The prevention of mine explosions, report and recommendations, by Victor Watteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

TECHNICAL PAPER 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

TECHNICAL PAPER 28. Ignition of mine gas by standard incandescent lamps, by H. H. Clark. 1912. 6 pp.

TECHNICAL PAPER 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

TECHNICAL PAPER 39. The inflammable gases in mine air, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 2 figs.

TECHNICAL PAPER 43. The effect of inert gases on inflammable gaseous mixtures, by J. K. Clement. 1913. 24 pp., 1 pl., 8 figs.

TECHNICAL PAPER 44. Safety electric switches for mines, by H. H. Clark. 1913. 8 pp.

TECHNICAL PAPER 47. Portable electric mine lamps, by H. H. Clark. 1913. 13 pp.

TECHNICAL PAPER 48. Coal-mine accidents in the United States, 1896-1912, with monthly statistics for 1912, compiled by F. W. Horton. 1913. 74 pp., 10 figs.

TECHNICAL PAPER 58. The action of acid mine water on the insulation of electric conductors; a preliminary report, by H. H. Clark and L. C. Ilsley. 1913. 26 pp., 1 fig.

TECHNICAL PAPER 75. Permissible electric lamps for miners, by H. H. Clark. 1914. 21 pp., 3 figs.

TECHNICAL PAPER 76. Notes on the sampling and analysis of coal, by A. C. Fieldner. 1914. 59 pp., 6 figs.

TECHNICAL PAPER 77. Report of the Committee on Resuscitation from Mine Gases, by W. B. Cannon, George W. Crile, Joseph Erlanger, Yandell Henderson, and S. T. Meltzer. 1914. 36 pp., 4 figs.

TECHNICAL PAPER 84. Methods of preventing and limiting explosions in coal mines, by G. S. Rice and L. M. Jones. 1915. 45 pp., 14 pls., 3 figs.

TECHNICAL PAPER 100. Permissible explosives tested prior to March 1, 1915, by S. P. Howell. 1915. 15 pp.

TECHNICAL PAPER 108. Shot firing in coal mines by electricity controlled from the outside, by H. H. Clark, N. V. Breth, and C. M. Means. 1915. 36 pp.

MINERS' CIRCULAR 5. Electrical accidents in mines, their causes and prevention, by H. H. Clark, W. D. Roberts, L. C. Ilsley, and H. F. Randolph. 1911. 10 pp., 3 pls.

MINERS' CIRCULAR 7. Use and misuse of explosives in coal mining, by J. J. Rutledge. 1914. 51 pp., 8 figs.

MINERS' CIRCULAR 8. First-aid instructions for miners, by M. W. Glasgow, W. A. Raudenbush, and C. O. Roberts. 1913. 67 pp., 51 figs.

MINERS' CIRCULAR 11. Accidents from mine cars and locomotives, by L. M. Jones. 1912. 16 pp.



MINERS' CIRCULAR 12. Use and care of miners' safety lamps, by J. W. Paul. 1913. 16 pp., 4 figs.

MINERS' CIRCULAR 14. Gases found in coal mines, by G. A. Burrell and F. M. Seibert. 1914. 23 pp.

MINERS' CIRCULAR 15. Rules for mine-rescue and first-aid field contests, by J. W. Paul. 1913. 12 pp.

MINERS' CIRCULAR 16. Hints on coal-mine ventilation, by J. J. Rutledge. 1914. 22 pp.

MINERS' CIRCULAR 21. What a miner can do to prevent explosions of gas and coal dust, by G. S. Rice. 1915. 24 pp.

# INDEX.

## A.

	Page.
Accidents, mine, at Lodge Mill Colliery.....	11-12
in Cripple Creek district.....	51
Acetylene, proportion of, in explosive mix- tures.....	31
Acetylene flame, extinguishing of, by decrease of oxygen.....	28, 30, 80
Acetylene lamp, use in detecting mine gas...	61
Afterdamp, detection of.....	14
Air, at sea level, percentage of oxygen in....	10
fresh, a necessity for miners.....	18
in coal mines, analyses of. 26, 27, 28, 33-41, 44-47	
black damp in, percentage of.....	48-49
carbon dioxide in, percentage of.....	27, 48
composition of, changes in.....	5, 22
leakage of, effect of.....	50
oxygen in, lack of.....	28
percentage of.....	48, 49, 50
<i>See also</i> Black damp.	
in metal mines, analyses of.....	63-78
black damp in, percentage of.....	66-76, 77
mine, changes in, causes of.....	78
explosibility of, effect of oxygen on..	80
humidity of, effect of.....	49
inflammability of, results of tests....	23
purity of.....	5
water vapor in, absorption of.....	6
pure, analyses of.....	6, 32
Alcohol flame, effect of extinguishing atmos- pheres on.....	30
Anaconda mine, description of.....	53
gas in, analyses of.....	53, 56, 60
sampling of.....	53
lights in, effect of oxygen on.....	62
Analyses, of black damp.....	33, 48, 77-78, 80, 81
in metal mines.....	77-78
of coal-mine air.....	26, 33-41, 44-47
of gas, from Anaconda mine.....	53, 56, 60
from Cresson mine.....	54, 57, 60
from Mary McKinney mine....	56-57, 58, 60
from Midget mine.....	55, 60
of inflammable mixtures.....	23
of metal-mine air.....	63-78
of pure air.....	6, 32
Ash sawdust. <i>See</i> Sawdust.	

## B.

Barometric pressure, effect of, on outflow of strata gas.....	52
oxygen pressure corresponding to.....	10
Bert, Paul, law on physiological action of a gas.....	9
on dangers from low barometric pressure..	9
Bibliography on absorption of gases by coal..	19

## Page.

Black damp, analyses of.....	33, 48, 77-78, 80, 81
in metal mines, composition of.....	77-78, 81
percentage of.....	77-78, 81
deaths from.....	12
definition of.....	5, 78
effect of, on lights.....	80
on men and animals.....	80-81
on methane-air mixtures.....	80
formation of.....	6, 26, 42
in mine air, percentage of..	48-49, 66-71, 76, 77-78
inflammability of, results of tests.....	23
specific gravity of, variation of.....	26, 80
term not synonym for carbon dioxide...	81
<i>See also</i> Carbon dioxide; Nitrogen; Oxy- gen.	
Blasting, in metal mines, effect on air.....	76, 77
Breathing, effect on, by decrease of oxygen..	9-14,
17, 78	
of carbon dioxide, effect of.....	7-9, 78
<i>See also</i> Hyperpnea.	
Butte, Mont., mines at, analyses of air in....	66-69

## C.

Cadman, John, on ventilation in English coal mines.....	15
Canaries, effect of oxygen lack on.....	12-14, 79
use of, in exploring mines.....	14
Candle flame, extinguishing of, tests.....	28, 30, 62
use of, in detecting mine gas.....	61
Carbide flame, extinguishing of.....	62
Carbon dioxide, action of, on methane-air mixtures.....	24
effect of, in ventilation systems.....	15
on men and lights.....	29, 61, 78
in black damp, percentage of.....	42,
47, 48, 49, 76, 77, 80, 81	
in coal mines, amount given off.....	25
increase of.....	19, 79
percentage of.....	43, 48, 49, 80
sources of.....	6, 16-17, 25, 79
in Cripple Creek mines.....	61
in English coal mines, percentage of....	79
in lungs, regulation of.....	7-8
in mixtures extinguishing flame, percent- age of.....	30
poisoning from.....	7
presence of, sign of.....	16
proportion of, in normal mine air.....	27
retention of, by coal.....	19
term not synonym for black damp.....	81
transfer of, in breathing.....	7
Carbon monoxide, in explosive mixtures, effect of.....	31

	Page.		Page.
Carbon monoxide flame, effect of extinguishing atmospheres on.....	30	Fires, mine, effect of sealing on.....	31-32
Carbon monoxide poisoning, effect of.....	12	oxygen in air, effect on.....	31-32
Clement, J. K., on inflammable mixtures containing oxygen.....	24-25	prevention of.....	19
on inflammability of methane-air mixtures.....	24-25	Flame, extinguishing of.....	28-31
Coal, absorption of gases by, bibliography...	19	Flax, flame of, effect of extinguishing atmospheres on.....	30
absorption of oxygen by, amount of.....	79		G.
power of.....	6	Gas, analyses of, from Anaconda mine....	53, 56, 60
oxidation of, experiments on.....	20	from Cresson mine.....	54, 57, 60
rate of.....	21	from Mary McKinney mine....	56-57, 58, 60
Coal dust, absorption of oxygen by.....	21	from Midget mine.....	55, 60
Coal-dust explosions, ventilation cause of....	18	combustion of.....	62-63
Coal-gas flame, effect of extinguishing atmospheres on.....	30	in rock strata, Cripple Creek mining district.....	51-63, 81
Coal mines, analyses of air in... 26, 27, 28, 33-41, 44-47		Gas blowers, in Cripple Creek district.....	54
black damp in, amount produced in....	78	Gas sampling, in Anaconda mine.....	53
carbon dioxide in, increase of.....	79	in Cresson mine.....	54
percentage of.....	80	in Cripple Creek mines.....	52-63
source of.....	6, 16-17, 25, 79	in Mary McKinney mine.....	58
oxidation of air in.....	77	in Midget mine.....	52, 53
oxygen in, English law relative to.....	17	Gases, absorption of, by coal, bibliography on.....	19
reduction of.....	17, 79	expansion of, effect of altitude on.....	10
<i>See also</i> English coal mines; Indiana; Pennsylvania.		inflammability of, limits of.....	24-25
Combustion, rate of, effect of oxygen content on.....	80	Gases, mine, analyses of.....	27, 28
Combustion, spontaneous, prevention of....	19	effect of atmospheric pressure on....	52
Conumdrum mine, ventilation in.....	53	explosibility of.....	23
Cresson mine, compressed air in, effect of....	54	stratification of.....	27
gas in, analyses of.....	54, 57, 60	<i>See also</i> Black damp; Carbon dioxide; Carbon monoxide; Nitrogen.	
sampling of.....	54	physiological action of, law on.....	9
lights in, effect of oxygen on.....	62	Gasoline flame, effect of extinguishing atmospheres on.....	30
location.....	53-54	Gasoline vapor, in explosive mixtures, effect of.....	31
Cripple Creek mining district, gas in rock strata in.....	51-63, 81	Graham, J. I., on oxidation of coal.....	20
Cypress shavings. <i>See</i> Sawdust.		on oxygen absorption by coal dust.....	21
	D.		H.
Douglas, C. G., on carbon dioxide, effect of, on breathing.....	8	Haldane, J. S., on deaths from black damp..	12
	E.	on effect of carbon dioxide on breathing..	7-8
English coal mines, carbon dioxide in, percentage of.....	79	on inflammable mixtures.....	23
oxygen in, law on.....	17	on light of safety lamp, in air low in oxygen.....	28
Ethyl-alcohol flame, effect of extinguishing atmospheres on.....	30	on lights in air low in oxygen.....	28
Ethylene, in explosive mixtures, effect of..	31	recommendation of, for standard of ventilation.....	16
Ethylene flame, effect of extinguishing atmospheres on.....	30	Hemlock sawdust. <i>See</i> Sawdust.	
Explosibility of methane, effect of diminished oxygen on.....	79-80	Hempel explosion pipette, use of, in inflammability tests.....	24
limits of.....	24-25	Henderson, Yandell, on breathing with decreased oxygen supply.....	11
Explosibility of methane-air mixtures, effect of black damp on.....	80	on effect of carbon dioxide on breathing...	8
effect of oxygen content on.....	79-80	Huddersfield, England. <i>See</i> Lodge Mill Colliery.	
experiments on.....	23-25	Humidity, effect of, on mine air.....	49
risk from, reduction of.....	23-25	Hydrogen, in explosive mixtures, effect of....	31
Explosibility of mine air, effect of oxygen on..	80	Hydrogen flame, effect of extinguishing atmospheres on.....	30
Explosions, mine, ventilation cause of.....	18	Hyperpnœa, causes of.....	9
Explosive mixtures of gas and air, oxygen in. <i>See also</i> Methane-air mixtures.	31		I.
	F.	Inflammable gas, in coal mines, danger of....	50
Fire damp, inflammability of, results of tests of.....	23	Inflammable mixtures, investigations of....	23-25
oxygen reduced by.....	12	with little oxygen, analyses of.....	23
		<i>See also</i> Methane-air mixtures.	



	Page.
Inflammability of fire damp, results of tests of.....	23
Inflammability of gases, limits of.....	24-25
Inflammability of methane.....	5-6
Inflammability of methane-air mixtures, limits of.....	24-25
Inflammability of mine air, effect of oxygen on.....	80
results of tests of.....	23
Indiana, coal mines, analyses of air in.....	46-47
J.	
Jorissen, W. P., on extinction of flames.....	30-31
K.	
Kellogg, Mont., mine at, analyses of air in.....	65-66
Kerosene flame, effect of extinguishing atmospheres on.....	30
L.	
Laws. <i>See</i> English coal mines; Pennsylvania.	
Lead, S. Dak., mine at, analyses of air in.....	70-71
Leakage of air, in mines, effect of.....	50
Lights, carbon dioxide given off by.....	25
extinguishing of.....	28, 30, 61, 62, 80
oxygen consumed by.....	25
Lingren, W., on gas in Cripple Creek mining district.....	51
Lodge Mill Colliery, black damp in, accidents from.....	11-12
M.	
Mahler, M. P., on absorption of oxygen by coal.....	20
Mary McKinney mine, air in, effect of wind and temperature on.....	59
gas in, analyses of.....	56-57, 58, 60
sampling of.....	58
lights in, effect of oxygen content on.....	62
location of.....	58
Metal mines, air in, analyses of.....	63-78
black damp in, average composition of.....	81
percentage of.....	66-76, 77-78, 81
blasting in, effect on air in.....	76, 77
carbon dioxide given off in.....	26
oxidation of air in.....	77
oxygen in, consumption of.....	26
Methane, effect of, on oxygen content of mine air.....	5-6
explosibility of, effect of lack of oxygen on.....	79-80
limits of.....	24-25
in coal mines, danger from.....	50
proportion of.....	43
in explosive mixtures, proportion of, effect of.....	31
inflammability of.....	5-6
Methane-air mixtures, explosibility of, effect of black damp on.....	80
effect of oxygen content on.....	79-80
experiments on.....	23-25
risk from, reduction of.....	18
inflammability of, limits of.....	24-25
<i>See also</i> Carbon dioxide.	
Methane flame, effect of extinguishing atmospheres on.....	30
Methylated-spirit flame, effect of extinguishing atmospheres on.....	30

	Page.
Mice, effect of low-oxygen atmosphere on.....	12-14, 79
use of, in exploring mines.....	14
Midget mine, description of.....	52
gas in, analyses of.....	55, 60
sampling of.....	52, 53
lights in, effect of oxygen content on.....	62
ventilation system in.....	52
Mine air. <i>See</i> Air, in coal mines; Air, in metal mines; Air, mine; Black damp; Gases, mine.	
Mine explosions. <i>See</i> Explosions, mine.	
Mine fires. <i>See</i> Fires, mine.	
Mine timber. <i>See</i> Timber, mine.	
Mines. <i>See</i> Coal mines; Metal mines.	
Moisture, influence of, on absorption of oxygen by coal.....	2-21
N.	
Natural gas, in explosive mixtures, effect of.....	31
Natural-gas flame, effect of extinguishing atmospheres on.....	30
Nitrogen, in black damp, percentage of.....	48, 49, 80, 81
in Cripple Creek mines.....	61
O.	
Oak sawdust. <i>See</i> Sawdust.	
Osborne, O. O., recommendation for standard of ventilation.....	16
Ovitz, F. K., on absorption of oxygen by coal.....	21
Oxidation of air, in coal mines.....	77
in metal mines.....	77
Oxidation of coal, experiments on.....	20, 21
rate of.....	20, 21
Oxygen, absorption of, by coal, amount of.....	79
mode of.....	19, 21
power of.....	6
utilization of.....	19
absorption of, by coal dust.....	21
consumption of, by lights in mines.....	25
decrease of, effect on lights.....	28, 30, 61, 62, 80
effect of, on breathing.....	9, 10-14, 61, 78-79
on combustion.....	80
resulting in more nitrogen.....	80
effect of, subordinated in ventilation systems.....	15
in air at sea level, percentage of.....	10
in black damp, percentage of.....	76, 77
in breathing, transfer of.....	7
in coal mines, English law on.....	17
reduction of.....	17, 19, 79
in gas mixtures, percentage of.....	28, 30, 31
in mine air, effect of, on fire.....	31-32
factors decreasing.....	6
percentage of.....	48, 49, 50
pressures of, corresponding to barometric pressures.....	10
<i>See also</i> Fire damp.	
P.	
Pennsylvania, anthracite coal mine in, analyses of air in.....	44-45
bituminous-mine law of.....	50
bituminous mines in, ventilation in.....	18
Petrol spirit flame, effect of extinguishing atmospheres on.....	30
Petroleum flame, effect of extinguishing atmospheres on.....	30

	Page.		Page.
Porter, H. C., on absorption of oxygen by coal.....	21	Timber, mine, decay of, effect of, on air .....	26
on oxidation of coal.....	21	decrease of oxygen content by.....	21-23
Priestly, J. G., on effect of carbon dioxide on breathing.....	8	increase of carbon dioxide by.....	21-23
Propagation of flame, importance of.....	23	source of black damp.....	23
R.		Tonopah mines, analyses of air in. 64-65, 69-70, 73-75	
Ralston, O. C., on oxidation of coal.....	21	Trojan, S. Dak., mines at, analyses of air in. 69	
Ransom, F. L., on gas in Cripple Creek mining district.....	51	V.	
Respiration. <i>See</i> Breathing.		Ventilation, difference in, in buildings and in mines.....	17-18
S.		in coal mines, necessity of.....	18, 50-51
Sampling of gas. <i>See</i> Gas sampling.		regulation of.....	15, 50, 51, 79
Sawdust, effect of, on composition of atmospheric air.....	22	in Conundrum mine.....	53
Schneider, E. C., on effect of carbon dioxide on breathing.....	8	in Midget mine.....	52
Sealing of mines, effect of, on fire.....	31-32	in Pennsylvania coal mines.....	18
Smith, Lorain, on effect of carbon dioxide on breathing.....	7	standard of.....	16
Specific gravity of black damp, variation of..	26, 80	Veteran, Nev., mine at, analyses of air in....	71-73
Spontaneous combustion, in mines, prevention of.....	19	W.	
T.		Water vapor, absorption of, by mine air.....	6
Temperature, effect of, on mine air....	48, 76-77, 80	determination of.....	6
on perspiration.....	15	West Virginia, coal mines in, analyses of air in.....	44-45
factor in determining water-vapor content.....	6	Wet-bulb temperature. <i>See</i> Temperature, wet bulb.	
wet-bulb, effect of, on mine air.....	76, 77, 80	Whalley, E. B., on ventilation in English coal mines.....	15
variation of.....	43	Winmill, T. F., on absorption of carbon dioxide by coal.....	20
		on oxidation of coal.....	20
		Wolf lamps, experiments with, in air low in oxygen.....	28
		Wood, effect of, on composition of air.....	22

DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

THE TECHNOLOGY OF MARBLE  
QUARRYING

BY

OLIVER BOWLES



WASHINGTON  
GOVERNMENT PRINTING OFFICE  
1916



The Bureau of Mines, in carrying out one of the provisions of its organic act—to disseminate information concerning investigations made—prints a limited free edition of each of its publications.

When this edition is exhausted copies may be obtained at cost price only through the Superintendent of Documents, Government Printing Office, Washington, D. C.

The Superintendent of Documents *is not an official of the Bureau of Mines*. His is an entirely separate office and he should be addressed:

SUPERINTENDENT OF DOCUMENTS,  
Government Printing Office,  
Washington, D. C.

The general law under which publications are distributed prohibits the giving of more than one copy of a publication to one person. The price of this publication is 30 cents.

*First edition. March, 1916.*

## CONTENTS.

---

	Page.
Preface, by CHARLES L. PARSONS.....	3
Marble in general.....	5
Definition of marble.....	5
Composition of marble.....	5
Origin of marble.....	6
Formation of the original limestone.....	6
Metamorphism of limestone.....	7
Origin of onyx marbles.....	7
Origin of verd antique.....	7
Physical properties of marble.....	8
Hardness.....	8
Specific gravity and weight per cubic foot.....	9
Solubility.....	11
Color.....	11
Translucence.....	13
Texture.....	13
Rift or grain.....	14
Porosity.....	14
Strength.....	16
Crushing strength.....	17
Transverse strength.....	17
Elasticity.....	17
Shearing.....	18
Weathering of marble.....	18
Varieties of marble.....	20
Distribution and production of marble in the United States.....	21
The imperfections of marble.....	22
Unsoundness.....	22
Meaning of unsoundness.....	22
Importance of joints in marble deposits.....	22
Nature of joints.....	22
Origin of joints.....	23
Practical illustrations of joint systems.....	26
Persistence of joints at depth.....	27
Unsoundness in verd antique.....	29
"Glass seams".....	29
Iron sulphides.....	29
The common iron sulphides.....	29
Manner and effect of sulphide decomposition.....	30
Sulphides not always injurious.....	30
Researches of Julien.....	31
Researches of Stokes.....	32
Methods of determining stability of sulphides.....	32
The uses of sulphide-bearing marbles.....	33

The imperfections of marble—Continued.	Page.
Silica.....	33
Origin of silica in marble deposits.....	33
Detrimental effects of silica in marble.....	34
Silicated marbles.....	34
Dolomite in marble.....	35
Undesirable colors.....	35
Fissility.....	36
Defects in texture and state of aggregation.....	37
Prospecting and developing marble deposits.....	39
Prospecting.....	39
Value of geologic maps.....	39
Need of detailed prospecting.....	39
Determination of overburden.....	39
Surface study insufficient.....	40
Diamond-drill prospecting.....	40
Mechanism of the double core-barrel drill.....	41
Value of double core-barrel drill to prospector.....	41
Drilling the holes.....	42
Numerous holes undesirable.....	42
Prospecting thick beds dipping at a moderate angle.....	42
Determination of dip and strike from drill cores.....	42
Arrangement of holes to test variation in quality.....	43
Arrangement of holes to determine unsoundness.....	43
Arrangement of the cores obtained.....	45
Value of the information gained from cores.....	45
Cost of marble prospecting.....	45
Preservation of cores.....	46
Stripping.....	46
Ordinary methods of stripping.....	46
Insufficient removal of stripping.....	46
Waste heaps impeding future development.....	47
Use of overhead cableway hoists.....	47
Hydraulic stripping.....	47
General plan of quarrying.....	48
Production of uniform grades of marble desirable.....	48
Factors governing plan of development.....	48
Effect of attitude of beds on plan of quarrying.....	48
Effect of overburden on plan of quarrying.....	49
Effect of uniformity of product on plan of quarrying.....	49
Quarry operations and equipment.....	50
Power plants.....	50
Types employed.....	50
Advantages of electricity.....	50
Source of electric power.....	51
Compressed-air and steam equipment.....	52
Channeling.....	53
Channeling machines.....	53
Channeling in relation to bedding.....	54
Influence of attitude of marble beds on channeling.....	54
Level-floor channeling.....	55
Saw-tooth floor channeling.....	55
Inclined-track channeling.....	56
High efficiency of inclined-track channeling.....	57

## Quarry operations and equipment—Continued.

## Channeling—Continued.

	Page.
Channeling in relation to bedding—Continued.	
Inclined-track channeling across beds.....	58
Influence of rift or color bands on channeling.....	59
Waste due to acute-angled blocks.....	59
Example of inclined channeling.....	60
Influence of dip and strike on direction of channel cuts.....	61
Channeling in relation to unsoundness.....	61
Channeling parallel with joint systems.....	61
Channeling coincident with joints.....	62
Economy of channeling in accordance with unsoundness.....	63
Utilization of joints for making cross breaks.....	64
An illustration of channeling in accordance with unsoundness....	64
A proposed improvement in quarry methods.....	65
Indistinct joints and joint systems.....	65
How to deal with complex jointing.....	65
Cutting with unsoundness in Tennessee quarries.....	68
Possible objections discussed.....	68
Methods of searching for indistinct joints.....	69
The boulder quarries of Tennessee.....	69
Study of unsoundness important.....	70
The quarry operator's prime object.....	70
Reduction in number of corner cuts.....	70
Channeling in relation to rift.....	71
Use of wire saw in quarries.....	71
Mechanism of the wire saw.....	71
The penetrating pulley.....	71
Instance of use of wire saw in quarry.....	72
Desirability of using long wire.....	73
Method of replacing an old wire by a new one.....	73
Disadvantages in use of wire saw.....	73
Drilling.....	74
Machinery.....	74
Drilling in relation to channeling.....	75
Arrangement and spacing of holes for cross breaks.....	76
Drill holes for floor breaks.....	76
Proper size for drill holes.....	76
Advantage of reaming drill holes.....	76
Sharpening and tempering of drills.....	77
Wedging.....	77
Types of wedges employed.....	77
Uniform strain desired.....	78
Effect of rift on drilling and wedging.....	79
Removal of key blocks.....	80
Removal of key blocks as waste.....	80
Making the floor break.....	81
Hoisting out key blocks.....	81
Hoisting.....	82
Turning down blocks.....	82
Cable attachment.....	82
Efficient handling of material.....	83
Tunneling.....	84
Definition of tunnel.....	84



## Quarry operations and equipment—Continued.

## Tunneling—Continued.

	Page.
General considerations.....	84
Opening tunnels across beds.....	84
Method of opening tunnels parallel with open beds.....	85
Method of quarrying on tunnel floor.....	86
Roof supports.....	86
Transportation of material in tunnels.....	87
Lighting and ventilation of tunnels.....	87
Undercutting.....	87
Combined tunnel and undercut.....	88

Drainage.....	88
---------------	----

General considerations in quarrying.....	88
--	----

Safety first.....	89
-------------------	----

Hoisting equipment.....	89
-------------------------	----

Types of derricks.....	89
------------------------	----

Use of gantry cranes.....	90
---------------------------	----

Derrick guys.....	90
-------------------	----

Guy anchors.....	90
------------------	----

Hoists.....	91
-------------	----

Location and size of derricks.....	91
------------------------------------	----

Hoist signaling.....	92
----------------------	----

Scabbling.....	92
----------------	----

Transportation of quarried rock to the mill.....	93
--	----

Various methods of haulage employed.....	93
--	----

Teams and wagons.....	93
-----------------------	----

Connecting quarries with railroad tracks.....	93
---	----

Cable cars.....	93
-----------------	----

Factors controlling the method of transportation.....	94
---	----

Equipment and operations in mills and shops.....	94
--	----

Location of plant.....	94
------------------------	----

Buildings.....	95
----------------	----

Heating and ventilation.....	95
------------------------------	----

Power.....	96
------------	----

Method of power transmission.....	96
-----------------------------------	----

Plan of mill, shop, and yard.....	97
-----------------------------------	----

Mill.....	98
-----------	----

Saw gangs.....	98
----------------	----

Abrasives used in sawing.....	98
-------------------------------	----

Sand pumps.....	99
-----------------	----

Sawing with unsoundness.....	99
------------------------------	----

Sawing corner blocks.....	99
---------------------------	----

Loading saw beds.....	99
-----------------------	----

The gang car.....	100
-------------------	-----

The transfer car.....	100
-----------------------	-----

An example of a well-equipped plant.....	101
--	-----

Unloading saw beds.....	101
-------------------------	-----

A convenient device on cable slings.....	102
--	-----

The wire saw.....	102
-------------------	-----

Shop or finishing plant.....	102
------------------------------	-----

Location of shop.....	102
-----------------------	-----

Coping.....	102
-------------	-----

Rubbing.....	103
--------------	-----

Gritting and buffing.....	103
---------------------------	-----

## Equipment and operations in mills and shops—Continued.

## Plan of mill, shop, and yard—Continued.

## Shop or finishing plant—Continued.

	Page.
Cubic stock.....	104
Planers.....	104
Carborundum machines.....	105
Column cutting.....	106
Cutting and carving.....	107
Handling material.....	107
Crating and shipping.....	107
Adequate water supply.....	107
The problem of waste.....	108
The importance of waste.....	108
Elimination vastly more important than utilization.....	108
Waste elimination.....	108
Unavoidable losses.....	108
Avoidable losses.....	109
Chief causes of waste.....	109
Systematic prospecting.....	109
Direction of quarry walls.....	109
Tunneling to avoid upper inferior beds.....	110
Waste in tunneling.....	110
Waste due to unsoundness.....	111
Waste due to lack of uniformity.....	111
Waste due to irregular blocks.....	111
The various regular forms of quarried blocks.....	112
Circumstances governing the production of the various forms.....	113
Waste due to acute-angled blocks.....	115
Justification for acute-angled blocks.....	115
General rules governing shape of blocks.....	117
Impurities that cause waste.....	117
Silica.....	117
Dolomite.....	118
Pyrite and marcasite.....	118
Mica.....	118
Waste due to bad color.....	118
Waste due to strain breaks.....	119
Waste utilization.....	119
By-products.....	119
Riprap.....	120
The ball breaker.....	120
Lime.....	120
Flux.....	120
Improvement of soils.....	121
Terrazzo.....	122
Cement.....	122
Road material.....	122
Rubble.....	122
Miscellaneous.....	122
Strain breaks in quarries.....	123
Losses due to strain.....	123
Strain causes waste, not impaired quality.....	123
Extent and importance of strain.....	123
Examples of strain phenomena recorded in literature.....	124

Strain breaks in quarries—Continued.	Page.
Examples of strain phenomena observed in field work. ....	129
General analysis of strain breaks recorded. ....	130
Relation of earthquakes to strain breaks. ....	131
Origin of the stresses. ....	132
Action of rocks under stresses. ....	133
The limitations of strain. ....	135
Effect of quarrying on strain. ....	135
Basic principles looking to reduction of damage from strains. ....	136
The remedy. ....	137
A successful experiment. ....	139
Cost keeping. ....	144
Bearing of cost keeping on selection of methods and machines. ....	144
Commercial records. ....	144
Technical records. ....	144
Relation of technical records to efficiency. ....	144
Qualifications of superintendents. ....	146
Real purpose of technical accounts. ....	147
The cost of not keeping accounts. ....	147
Localization of costs. ....	147
General and special accounts. ....	148
General accounts affecting the whole plant. ....	149
General expense. ....	149
Land factor. ....	150
Organization factor. ....	150
Yard-operation factor. ....	150
Power factor. ....	151
General quarry accounts. ....	153
Direct quarry cost. ....	153
Overhead expense. ....	153
Total cost of operating quarry. ....	154
Quarry production account. ....	155
Condensed cost account. ....	155
Special quarry accounts. ....	156
General mill accounts. ....	159
Direct mill cost. ....	159
Overhead expense. ....	160
Total cost of operating mill. ....	161
Unit cost account. ....	161
Condensed cost account. ....	162
Special mill accounts. ....	163
Shop or finishing plant accounts. ....	165
Job accounts. ....	167
Publications on mineral technology. ....	168
Index. ....	171

## ILLUSTRATIONS.

PLATE I. <i>A</i> , The abrupt termination of joints in a plastic layer in which deformation by flowage takes the place of fractures; <i>B</i> , A system of parallel joints emphasized by erosion.....	Page. 26
II. Double core-barrel drill.....	42
III. <i>A</i> , Serviceable car for removal of waste and stripping; <i>B</i> , Typical example of steam leakage in a quarry transmission line; <i>C</i> , Track supported in level position for "saw-tooth" floor channeling.....	46
IV. Sullivan "Duplex" channeler.....	54
V. <i>A</i> , A quarry floor suitable for inclined-track channeling; <i>B</i> , Method of quarrying acute-angled blocks; <i>C</i> , Channeling diagonally on a quarry floor inclined 36°.....	58
VI. <i>A</i> , Cable sling for hoisting blocks of stone; <i>B</i> , Undercutting parallel with steeply inclined beds in quarry at West Stockbridge, Mass.; <i>C</i> , Ball breaker used in quarry at Gantts Quarry, Ala.....	84
VII. <i>A</i> , A tunnel in a Vermont marble quarry; <i>B</i> , Gantry crane employed at quarry at Ashley Falls, Mass.; <i>C</i> , Manner in which open seams pitch into the corner of a quarry when the quarry wall meets the strike obliquely.....	86
VIII. <i>A</i> , A guy anchor post used in Tennessee; <i>B</i> , An angle steel and concrete guy anchor used at quarry at Gantts Quarry, Ala.....	92
IX. Carborundum machine cutting a molding in marble.....	106
X. <i>A</i> , A carborundum machine adapted for cutting curved moldings; <i>B</i> , A carborundum machine adapted for cutting balusters.....	106
XI. A method of channeling right-angled blocks on an inclined floor.....	114
XII. <i>A</i> , Method of quarrying monoclinic blocks; <i>B</i> , Vertical drill holes partly closed by rock expansion.....	114
FIGURE 1. Diagrams illustrating the manner in which a single force may produce several systems of joints.....	25
2. A remarkable system of 21 parallel joints, as found in one outcrop..	26
3. Open joints in a quarry wall.....	27
4. Surface plan of joints that appeared to lack systematic arrangement.....	28
5. Proper positions of channeling machine tracks for beds of different inclinations.....	55
6. A "saw-tooth" quarry floor.....	56
7. Manner in which an inclined floor may be developed from a "saw-tooth" floor.....	57
8. Proper positions of channeling machine tracks for beds of different inclinations.....	59
9. The right and the wrong way of channeling unsound marble.....	62
10. The economy of channeling in accordance with joint systems.....	63
11. Diagram of joint systems in one bed of an Alabama marble quarry..	67



	Page.
FIGURE 12. Method of channeling a quarry floor employed by a Georgia marble company.....	70
13. Details of wedge that is effective in a small drill hole.....	78
14. Influence of rift on bottom breaks.....	79
15. Lewis pin used in hoisting key blocks.....	82
16. Method of driving a tunnel in marble.....	85
17. Present and proposed arrangement of pillars in a Colorado quarry.....	87
18. Track arrangement at a Vermont marble mill.....	97
19. Plan of mill and yard of a Maryland marble company.....	98
20. Improved method employed by one marble company for cutting corner blocks.....	100
21. Iron plate with ball bearings used to facilitate movement of marble blocks.....	106
22. Right-angled block.....	112
23. Monoclinic block.....	112
24. Oblique block.....	113
25. Triclinic block.....	113
26. Cuts by which blocks are separated from the solid mass.....	114
27. Diagram showing waste resulting from cutting monoclinic blocks into cubic stock.....	115
28. Diagram showing waste resulting from cutting an oblique block into cubic stock.....	116
29. Diagram showing small amount of waste resulting from cutting monoclinic blocks into thin stock.....	117
30. Method of drilling holes to avoid strain breaks.....	139
31. Plan of part of a marble quarry showing position of a rock mass studied.....	140
32. Appearance of drill holes and channel cut after expansion of rock.....	141
33. Second and third drill holes from the left end of the line shown in figure 32, illustrating the manner in which the holes were closed by compression.....	142

## PREFACE.

---

In 1914 an agreement was entered into, between the United States Geological Survey, the United States Bureau of Standards, and the Bureau of Mines, for a cooperative study of the building-stone industry of the country. In general, this agreement provided that the Geological Survey should study the data, compare the classification, extent, and geology of undeveloped and quarried deposits of stone and collect statistical information on production and value of output; that the Bureau of Standards should undertake all the physical and chemical tests required for determining the value of stone for structural purposes, or as aggregate for concrete, and that the Bureau of Mines should investigate all mining and technologic data, with special reference to safety, efficiency of operation, and prevention of waste. The general object of the cooperative agreement was to obtain comprehensive data on the occurrence, quality, and methods of preparation for the market of the various building stones of the United States. By agreement the work was begun with an investigation of the marble-quarrying industry.

The Bureau of Mines was fortunate in procuring for its share of the work the services of Mr. Oliver Bowles who had had experience in examining and describing quarries in Minnesota.

During 1914 Mr. Bowles personally visited 64 active marble quarries. Through the cooperation of quarry owners numerous representative samples of the marbles of the United States were obtained and shipped to the Bureau of Standards for testing. The results of these tests will appear in due time. The description of the investigations of quarry methods and the general technology of marble are published herewith.

The aim of this bulletin is to present to the marble workers of the United States a concise statement of the most efficient and economical methods now in use for producing and preparing marble. Progressive quarrymen realize the value of observing the methods employed by others in similar work, and it is hoped that the summary here presented will prove of use in their operations and will save them the inconvenience and expense of extensive travel to obtain such information for themselves.

The observations outlined refer chiefly to the methods employed in the quarrying of marble and the conditions that affect successful

operation; the structure of marble and its relation to quarry processes; waste through rock imperfections and inefficient quarry methods; and means of eliminating such waste or utilizing it if elimination is impossible. No attempt has been made to describe in detail all apparatus and equipment used in quarry operations, but special apparatus and any improvements or special designs tending toward increased efficiency are dealt with particularly.

Mr. Bowles has pointed out especially the great need of better systems of cost keeping as a means of keeping check on all unnecessary expense and as a means of testing the efficiency of methods and machines used in quarry operations. The importance of this feature can not be unduly magnified.

There is also pointed out a method for relieving earth pressure in certain quarries particularly subject to natural strains. This method promises to eliminate many of the fractures that constitute a prolific source of waste in quarry operations.

Operators of structural limestone quarries will find that many of their problems are discussed in this bulletin, as the methods and machinery they employ are similar to those used in quarrying marble. A more extended investigation of the particular problems of structural limestone and all other phases of limestone quarrying is contemplated.

Acknowledgment is made of many helpful suggestions received during the preparation of this bulletin. Too much emphasis can not be placed upon the valuable assistance rendered by Maj. J. S. Sewell, general manager of the Alabama Marble Co. His technical training, combined with practical experience, fitted him peculiarly for supplying just such information as the bureau desired and this information he imparted freely.

Others to whom acknowledgment is due are George R. Taylor, Marble, Colo.; J. P. McCluskey, Gantts Quarry, Ala.; A. W. Edson, Proctor, Vt.; John Kern and B. L. Pease, Knoxville, Tenn.; Prof. C. H. Gordon, University of Tennessee; Prof. F. J. Alway, University Farm, St. Paul, Minn.

The bureau desires to express its appreciation of the cordial treatment extended by quarrymen to its representative and the spirit of cooperation almost universally in evidence.

CHARLES L. PARSONS,  
*Chief, Division of Mineral Technology.*

# THE TECHNOLOGY OF MARBLE QUARRYING.

---

By OLIVER BOWLES.

---

## MARBLE IN GENERAL.

### DEFINITION OF MARBLE.

In its geologic sense the term marble is applied to rocks consisting of crystallized grains of calcite or dolomite or a mixture of the two. Although limestone has the same chemical composition as marble it differs in that the component particles of calcium or magnesium carbonates are granular and noncrystalline. In marble the crystals may be intimately intergrown, whereas limestone is an aggregation of unrelated particles cemented together into a solid mass.

In its commercial sense the term marble has a much wider application. As susceptibility to polish is one of its chief commercial assets, all calcareous rocks capable of polish are classed as marbles. Limestones that show little crystalline structure may, if they take a good polish, be classed as marbles. Furthermore, serpentine rocks, even if they contain little calcium or magnesium carbonate, are classed as marbles, as they are commercial substitutes of true marbles.

### COMPOSITION OF MARBLE.

Aside from serpentine and other extraordinary varieties, marble is made up almost entirely of calcium or magnesium carbonates. A calcite marble may consist of 96 to 99 per cent calcium carbonate. A dolomite marble, if impurities are disregarded, contains approximately 54 per cent calcium carbonate and 46 per cent magnesium carbonate. Marbles consisting of mixtures of calcite and dolomite may have compositions anywhere between these two extremes. The extremes may be illustrated by examples mentioned by Dale.<sup>a</sup> Marble quarried near Proctor, Vt., contained 98.37 per cent calcium carbonate, and a dolomite marble from Lee, Mass., contained 54.05 per cent calcium carbonate and 45.93 per cent magnesium carbonate. An intermediate type is represented by the crystalline magnesium limestone of Tuckahoe, N. Y., which contained 70.1 per cent calcium carbonate and 25.40 per cent magnesium carbonate.<sup>b</sup>

---

<sup>a</sup> Dale, T. N., The commercial marbles of western Vermont; U. S. Geol. Survey Bull. 521, 1912, p. 13.

<sup>b</sup> Kemp, J. F., A handbook of rocks, 1906, p. 138.



A varying percentage of chemical impurities is present in practically all marbles. The more common of these are silica ( $\text{SiO}_2$ ), iron oxides ( $\text{FeO}$  and  $\text{Fe}_2\text{O}_3$ ), manganese oxide ( $\text{MnO}$ ), alumina ( $\text{Al}_2\text{O}_3$ ), and sulphur; less common are minute quantities of the oxides of titanium, potash, sodium, lithium, and phosphorus. Organic matter is commonly present.

The impurities of marble are present in the form of grains of definite minerals. In some specimens the individual grains may be too minute to be recognizable with the naked eye, and in others they may attain considerable size. The more common mineral impurities are quartz (or some other form of silica, such as chert or flint), hematite, limonite, graphite, mica, chlorite, tremolite, wollastonite, diopside, hornblende, tourmaline, pyrite, or marcasite. In the marbles of southern Ontario, Parks <sup>a</sup> notes the occurrence of 37 minerals that have been formed by metamorphic processes acting on the impurities of the original limestone. The more common are quartz or some other form of silicon dioxide, pyrite, marcasite, mica, or chlorite.

Most marbles of commercial value contain small percentages of impurities.

### ORIGIN OF MARBLE.

#### FORMATION OF THE ORIGINAL LIMESTONE.

Marble is derived from beds of limestone. The latter are formed in the sea, mainly as accumulations of calcareous remains of marine organisms, such as corals, rhizopods, and algæ. Water containing carbon dioxide is capable of dissolving calcium carbonate from the rocks through or over which it flows, and in consequence the water of rivers is charged with lime carbonate as it enters the ocean. Thus a supply of dissolved calcium carbonate is always at hand from which the organisms may manufacture their shells. Countless generations live and die and as a consequence the calcareous accumulation may be of vast extent. In places, the chemical precipitation of calcium carbonate may add to this accumulation.

There is abundant evidence that many limestones are of organic origin, as some of them are merely aggregates of fairly well preserved shells. In most specimens, however, a few fragmentary shells only remain in recognizable form, all others, through the beating of the waves or other activities, having been broken into minute fragments.

By pressure of superincumbent material and by deposition of some form of cement in the intergranular spaces the mass is later consolidated as a firm and coherent rock which is termed "limestone." Beds hundreds and even thousands of feet in thickness have been formed by such processes.

<sup>a</sup> Parks, W. A., Report on the building and ornamental stones of Canada, vol. 1, 1912, p. 307.

## METAMORPHISM OF LIMESTONE.

Marble is regarded as being the product of the metamorphism of limestone beds. That granular noncrystalline limestone can be changed into crystalline limestone or marble has been proven in the laboratory, as shown by Clarke.<sup>a</sup> From the results of various experiments he concludes that pressure alone, heat alone, or both together may result in the recrystallization. It is probable that the presence of water assists the process. Marble may therefore result from great pressure exerted on the strata by folding, or by heat produced from an igneous intrusion, or both agencies may work in conjunction. Recrystallization as a result of igneous intrusion has been observed by several authors.<sup>b</sup>

## ORIGIN OF ONYX MARBLES.

Onyx marbles have a history rather distinct from that of the true marbles. Although consisting essentially of calcium carbonate, they are purely chemical deposits and have not resulted from the metamorphism of preexisting limestone beds. As pointed out by Merrill,<sup>c</sup> who gives a lengthy discussion of their origin and occurrence, they are of two types. One is a product of precipitation from hot springs, a travertine; the other is a deposit from cold-water solutions in limestone caves. Most deposits of onyx are formed in successive layers. Impurities such as iron and manganese oxides may be present in varying amounts in successive layers, and thus a beautiful banding may result. From the nature of their formation onyx deposits are necessarily limited in extent as compared with deposits of true marbles.

## ORIGIN OF VERD ANTIQUE.

Verd antique or serpentine marble is in no respect comparable with true marble either in composition or in origin. Serpentine is in general derived from the alteration of basic igneous rocks such as the peridotites, which are rich in olivine and pyroxenes, or from magnesium silicate rocks formed by the metamorphism of limestone. The process is accompanied by hydration, with an addition of 13 to 14 per cent of water. The movement occasioned by the swelling that results probably accounts for most of the unsoundness common to verd antique.

<sup>a</sup> Clarke, F. W., The data of geochemistry: U. S. Geol. Survey Bull. 491, 1911, pp. 531-532.

<sup>b</sup> Renwick, W. G., Marble and marble working, 1909, p. 4; Conybeare, W., Descriptive notes on the north-east coast of Ireland. Trans. Geol. Soc. London, vol. 3, 1816, p. 210.

<sup>c</sup> Merrill, G. P., Stones for building and decoration, 1903, pp. 242-296.

## PHYSICAL PROPERTIES OF MARBLE.

## HARDNESS.

Hardness may be defined as the resistance that the surface of a substance offers to abrasion. The hardness of calcite is given as 3 in Moh's scale, and that of dolomite as 3.5 to 4, whereas that of glass is about 5. The hardness of a marble as a whole may be different from that of the individual grains that compose it. The hardness is influenced by the degree of cohesion between the grains. Most fine-grained, compact marbles are harder than coarse-grained varieties. Some marbles are remarkably hard even if no silica or other excessively hard impurities are present. Hardness of the mass as a whole is an indication of "workability," and is an important property, as the cost of quarrying marbles that are worked slowly by tools is much higher than the cost of quarrying those easily worked. Although the cost of quarrying hard marble may be high, the hardness is a valuable property if the material is to be exposed to abrasion.

High resistance to abrasion is desirable in marbles that are to be used for sills, steps, or floor tile, all of which are exposed to the friction of the feet of pedestrians. Marble employed for such uses should be hard, and uniformity in hardness is desirable; otherwise the surface will soon become uneven. In constructing floor patterns of different marbles, it is important that the several varieties should be equally resistant to abrasion, as otherwise the floor will eventually become uneven. This condition may be observed in the floor of the Union Station at Washington, D. C., where the white tiles of Vermont marble, after eight years' use, are in places worn down nearly half an inch lower than the small squares of harder material from Swanton, Vt.

A second agent of abrasion is wind. Wind polish of a pronounced character has been observed on rocks much harder than marble. The Sioux quartzite of southwestern Minnesota, a rock that is probably harder than any other in the United States used for building purposes, has been so wind worn that corners have been rounded and the exposed surfaces have been given a glassy polish. Dust and sand carried by the wind on city streets tend slowly to wear away surfaces, mainly by removing insecure grains and thus exposing fresh surfaces to the agencies of weathering. The effects are most pronounced on corners and in narrow spaces between buildings where the force of the wind is concentrated. Egleston <sup>a</sup> states that in New York City many tombstones that face the prevailing winds are so worn that inscriptions are almost illegible.

---

<sup>a</sup> Egleston, J., The cause and prevention of decay in building stones: *Am. Architect*, vol. 18, September 5, 1885, p. 113.



## SPECIFIC GRAVITY AND WEIGHT PER CUBIC FOOT.

The specific gravity of a substance is its weight compared with the weight of an equal volume of water. The specific gravity of calcite is 2.7 and that of dolomite about 2.9. Consequently, dolomite marbles are heavier than calcite marbles. It is found that the actual weight per cubic foot of a block of marble differs more or less from its theoretical weight calculated from the specific gravity of the constituent minerals. A porous rock of given volume will be lighter than an equal volume of similar material that is nonporous. In most marbles the pore space is small, and the actual weight does not differ greatly from that calculated from the specific gravity.

The specific gravity of a compact homogeneous substance having no pore space may be determined in a simple manner, as follows: A thoroughly dried specimen is suspended from a balance by a thread and weighed in the air. It is then weighed while immersed in water. On account of the buoying up of the water it will weigh less while immersed. The loss in weight when immersed is the weight of a quantity of water equal in volume to the substance immersed. The specific gravity is, therefore, the weight of the substance in air divided by its loss in weight when immersed in water, as expressed by the formula  $\frac{A}{A-B}$ , in which A represents the weight of the substance in air, and B represents the weight of the substance when immersed in water.

An accurate determination of the specific gravity of a marble is, however, a less simple matter, on account of the pore space involved. In order to determine specific gravity accurately the pore space must be eliminated. This may be accomplished by the following procedure: Dry the rock specimen at 110° C. until all the moisture has been driven from the pores; then determine the dry weight. Next completely fill the pores with water, as by soaking the blocks in water for several weeks. Buckley,<sup>a</sup> accomplished the desired result by boiling the specimens for 36 hours under the receiver of an air pump that reduced the pressure to one-twelfth of an atmosphere. By such means the removal of air from all the pores may be facilitated. With the pores thus filled with water, weigh the specimen when immersed in water and determine the specific gravity as described above.

A more accurate method if care is exercised is to eliminate the pores by grinding the rock to a fine powder, and to determine the specific gravity of the powder. This may be done by means of a specific gravity bottle, with an accurately ground stopper that projects upward as a hollow tube of small diameter. The bottle is

<sup>a</sup> Buckley, E. R., Building and ornamental stones of Wisconsin: Wisconsin Geol. and Nat. Hist. Survey Bull. 4, 1898, pp. 65-67.



dried carefully and weighed. It is then filled with distilled water exactly to the top of the capillary tube and weighed again. Ordinarily the next step is to thoroughly dry the bottle, place within it a part of the finely powdered rock, and weigh. The difference between this weight and that of the empty bottle is the weight of rock taken. The rock powder is retained in the bottle, which is then filled with water and weighed again.

Certain mechanical difficulties make desirable a modification of the last two steps. Considerable time is required to thoroughly dry the narrow-necked bottle. After the dry mineral has been weighed and the bottle filled with water, it is difficult to prevent small particles from floating to the surface and flowing away with the superfluous water when the stopper is inserted. This small error may be avoided by reversing the last two steps. When the bottle full of water is weighed, the bulk of the water may be thrown out, the rock powder poured in, and enough water added to completely immerse it. After all bubbles have been removed the bottle may be filled with water to the top of the capillary tube as before and weighed. The loss of any fine particles during the process of filling and inserting the stopper does not affect the result, as the weighing is done subsequently. After the weighing has been completed, if the powder is thoroughly settled, and the water above it clear, most of the water may be carefully drawn off with a pipette. The remainder may be evaporated in a hot air bath, and the drying continued until the weight is constant. Thus the weight of the bottle containing the dry rock powder is the last figure obtained.

#### EXAMPLE.

If X = weight of dry bottle  
 Y = weight of bottle full of water  
 Z = weight of bottle containing powdered rock and filled with water  
 and A = weight of dry bottle and powdered rock  
 then  $A - X = \text{the weight of the rock employed} = W$ .

The sum of W and Y gives the total weight of rock, bottle, and water when none of the water is displaced. When the mineral is placed in the bottle and its volume of water is displaced the weight Z results. Therefore the weight of water displaced by the rock is  $Y + W - Z = M$ . The specific gravity is therefore  $\frac{W}{M}$ .

Marbles range in actual weight from 165 to 180 pounds per cubic foot. The economic significance of weight is chiefly in connection with the necessary strength of equipment for handling and freight charges for transportation. By knowing the average weight per cubic foot the quarryman may measure a block and calculate its weight with reasonable accuracy. He is thus enabled to judge the risk involved in handling it with any given equipment. Marble is a heavy structural material, and the necessary transportation charges

must be carefully considered when bidding on contracts at a distance from the point of production.

#### SOLUBILITY.

Calcium and magnesium carbonates are practically insoluble in pure water. Certain dissolved gases, notably carbonic acid gas, which are present in surface water in small proportions, render the water capable of dissolving the carbonates to a limited degree. Marbles exposed to the weather are therefore slowly dissolved. Although the process is slow, its effects may be considerable when long periods of time are involved. That marbles are more soluble than rocks consisting of silicate minerals is demonstrated in nature. Most marble deposits in humid regions are found in valleys formed by the more rapid erosion of the marble belts than of the bordering siliceous rocks.

The rate of solution is variable in different marbles. It depends on the chemical composition, texture, and porosity of the marble, the climate of the region, and the nature of the atmosphere. Near large cities various acids from smoke abound in the air and are taken up by rain water, thus increasing its power of solution. If a rock is permeable it dissolves more rapidly than if impervious. Calcite dissolves more rapidly than dolomite under the same given conditions if the texture of each is similar, but the tendency for dolomite to occur with granular texture usually reverses the order of their solubility. The solubility of marble deserves careful consideration if its use for exterior purposes is contemplated.

#### COLOR.

The color of a marble is one of its most important physical properties. It is governed by the nature of the constituents. Marbles consisting of pure calcite or dolomite are white because these minerals are white. A serpentine marble is green because the prevailing mineral, serpentine, is green. Variations from the white color of a pure marble are due to admixtures of foreign substances. Such impurities may be uniformly distributed and thus give a uniform coloration, or they may be present in bands or streaks, giving clouded or otherwise nonuniform colors. Examples of nonuniform color distribution are the "Pocahontas" marble of Alabama, the variegated marbles of Vermont, and the "crow foot" structure, irregular dark lines, characteristic of the Tennessee deposits.

The causes of some colors in marbles are easily determined. The black and grayish shades are to be attributed to carbon, which is usually present as fine scales of graphite though amorphous in the "crow foot" of Tennessee marble. Red, pink, or reddish-brown

shades are due to the presence of manganese oxides ( $\text{MnO}$ ) and ( $\text{Mn}_2\text{O}_3$ ) or to hematite ( $\text{Fe}_2\text{O}_3$ ). Yellow-brown, yellow, or cream colors are caused by minute grains of the hydrous oxide of iron, limonite ( $2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$ ). Dale<sup>a</sup> attributes the green color of certain Vermont marbles to the presence of sericite (fibrous potash mica), and the purplish tint of one of the dolomites of the Lake Champlain region to a mixture of hematite and magnetite. Referring to some of the less common colors, Dale comments as follows:

The more uncommon colors of marble are purplish, as in the Pavonazzo and Seravezza breccias imported from Italy, bright yellow, as in the "Giallo Antico" from North Africa, and orange yellow, as in some marbles from Norway. Among the uncommon combinations of colors is that of rose-pink and deep green in the "Leifset Gloire" from Norway.

The same author states <sup>b</sup> that the dolomite of Hancock and Mount Tabor, Vt., owes its buff color to the presence of siderite. Vogt<sup>c</sup> attributes the sky-blue, red, and orange tints of some Norwegian marbles to organic compounds. The "golden vein" of the Colorado-Yule marble is thought to be caused by the permeation of solutions bearing manganese or iron oxides. The green bands in certain parts of the same quarry are, according to Merrill,<sup>d</sup> caused by the presence of chrome-mica. Parks<sup>e</sup> describes green marbles in Ontario in which the color is due to needles of light-green actinolite. The pink color of one marble he attributes to "scattered flakes of brownish glistening mica." He states that brown colors are in many instances due to the presence of mica. He refers also<sup>f</sup> to a marble having gray dots due in part to some crystals being clear and others milky, and in part to the presence of fine graphite. The clear crystals are dolomite, and the milky crystals calcite. Solution emphasizes the dotted effect.

For certain purposes, it is desirable to have a distinct contrast between chiseled and polished surfaces. Such a contrast is especially desirable in headstones on which inscriptions are cut. The contrast is usually more pronounced in the colored, and less marked in the white marbles. A chiseled surface is opaque and somewhat granular, and reflects rather than absorbs the light. Hence it tends to appear white or light colored even if the stone is dark. When a face is polished the reflecting surfaces are removed, permitting the light

<sup>a</sup> Dale, T. N., The commercial marbles of western Vermont: U. S. Geol. Survey Bull. 521, 1912, p. 20.

<sup>b</sup> Dale, T. N., The calcite marble and dolomite of eastern Vermont: U. S. Geol. Survey Bull. 539, 1915, p. 54.

<sup>c</sup> Vogt, J. H. L., Norsk marmor, Kristiania, 1897, p. 354.

<sup>d</sup> Merrill, G. P., A report on the Colorado-Yule marble properties, 1914, p. 19.

<sup>e</sup> Parks, W. A., Report on the building and ornamental stones of Canada, vol. 1, 1912, pp. 323-324.

<sup>f</sup> Parks, W. A., Op. cit., p. 329.



to enter the crystals and be absorbed, causing the polished surface to appear much darker than the chiseled surface.

#### TRANSLUCENCE.

Marbles differ greatly in their capacity for transmitting light. The more translucent varieties, if fine grained, are best adapted for novelties or other ornamental purposes. Some marbles are waxy in appearance, and this property seems to be related to translucence. Dale<sup>a</sup> gives the depths to which certain foreign marbles will admit light. They are as follows: Best Pentelicon, 0.59 inch; Parian, 1.37 inches; Carrara statuary, 1.18 to 1.57 inches. The reputation of some marbles depends greatly upon this quality. As far as is known no figures have been obtained for the depth of light penetration into American marbles. Certain beds of Alabama marble are notably translucent. The same quality has been observed in marbles from Massachusetts, Vermont, and Colorado. Certain modes of artificial treatment are known to increase the translucence of marble. Usually the effects of such treatment are far less permanent than the material itself, and consequently are not to be recommended.

#### TEXTURE.

The grains of calcite and dolomite that make up a marble mass are crystalline and have a definite rhombohedral cleavage. They are mostly twinned. Both the cleavage and the twinning of each grain are independent as relating to other grains. The texture is usually about the same in all directions, though in some marbles an elongation of grains in one direction has been noted. This characteristic is discussed more fully under the heading "Rift." The degree of interlocking of grains, and other features of cohesion have a definite relation to crushing strength, porosity, and workability. In certain dolomitic marbles the grains of dolomite may differ greatly in size and shape from those of calcite. The difference in grain diameter between the larger and smaller grains in some marbles is rather marked, and in others is small. Uniformity of grain is desirable.

The size of grain is commonly described as fine, medium, or coarse. Such terms are indefinite, and may have quite different meanings with different individuals, the interpretation being dependent upon the range of texture experienced by the observer. In order to place texture upon an absolute basis Dale<sup>b</sup> graded the marbles of Ver-

<sup>a</sup> Dale, T. N., The commercial marbles of western Vermont: U. S. Geol. Survey Bull. 521, 1912, p. 17.

<sup>b</sup> Dale, T. N., Op. cit., p 54.



mont into six classes, based upon the average grain diameter. The classification is shown in the following table:

*Classification of Vermont marbles by grades of texture.*

Grade.	Grain diameter.			
	Maximum.	Average.	General averages.	
			Millimeters.	Inches.
	<i>Millimeters.</i>	<i>Millimeters.</i>		
1, extra fine.....	0.2	0.05 to 0.10	0.06	0.0023
2, very fine.....	.5	.07 to .16	.10	.0039
3, fine.....	.75	.10 to .25	.12	.0047
4, medium.....	1.0	.12 to .31	.15	.0059
5, coarse.....	1.5	.20 to .60	.24	.0094
6, extra coarse.....	2.54	.30 to 1.35	.50	.0196

The texture of marble is influenced by impurities such as graphite, sericite, tremolite, actinolite, and mica; and also by the folding or plication of the beds. The latter may cause elongation of grain or granulation of certain parts of the mass.

#### RIFT OR GRAIN.

The terms rift and grain are used synonymously for the direction of easiest splitting in marble. The rift is usually parallel with the bedding. It is probably due to elongation of grain caused by pressure. Dale<sup>a</sup>, whose microscopic study of marbles has been extensive, states that "in some marbles one or two axes of the grains are much longer than the others, and the longer axes of different grains are parallel, giving the rock a certain schistosity which is usually parallel to the bedding."

The rift may be emphasized by the presence of fibrous or platy minerals such as scales of mica or graphite or needles of actinolite. These usually occupy positions with their long axes parallel to the direction of grain elongation, and thus increase the tendency to split in that direction.

Rift is a property of marble that the quarryman should take into account in planning operations. By taking advantage of this ease of splitting, drill holes for wedging may be spaced much farther apart than if no rift exists.

#### POROSITY.

Porosity is the volume of pore space expressed as a percentage of the total volume of the rock mass. The pore space of marbles is usually much less than that of limestones and sandstones, and more

<sup>a</sup> Dale, T. N., Op. cit., p. 18.

than that of granites, though exceptions toward both extremes are known. It varies from 0.0002 to 0.4 per cent.

A method of determining pore space may be quoted from Parks,<sup>a</sup> as follows:

The specific gravity is determined as already indicated and the test piece, full of water, is weighed. The difference between this weight and the dry weight of the sample gives the weight of the included water. If this latter figure be multiplied by the specific gravity of the stone, we obtain an expression which represents the weight of the stone which would be required to fill the pores. If this amount be now added to the weight of the dry sample, the result is the weight of that sample, provided that there were no pore spaces. This weight divided into the weight of the stone required to fill the pores, and multiplied by 100, gives the percentage of pore space in the stone.

The difficulty of filling all the pores by saturation has already been referred to in considering specific-gravity determinations. A method is here suggested by which this difficulty may be avoided.

Three factors are necessary in order to determine porosity. They are specific gravity, dry weight, and volume of the material employed. The specific gravity may be determined by the pycnometer (specific-gravity bottle) method as already described. The volume and the dry weight may be determined from a smooth-faced cube of about 1-inch edge. Drying the cube at 110° C. until the weight is constant gives the dry weight. The specimen may then be coated with a thin film of paraffin, wax, or other waterproof substance to make it impervious, and its volume may be determined from water displacement in a large specific-gravity bottle. The film must be made as thin as possible, as its presence increases the volume of the block. If the quantity of waterproof substance used in coating the block is determined, a correction may be made by subtracting its volume from the total volume obtained.

Another method is to saturate the block and then determine its volume by water displacement without any coating. If it is even approximately saturated, further absorption during the brief time occupied in determining its water displacement will be negligible.

The method of calculating porosity is as follows:

If X = dry weight of block

G = specific gravity of material composing block

and V = volume of block

Then  $V \times G$  = weight of block if all the pores were filled with marble = Y.

$Y - X$  = weight of marble required to fill pores = W

Percentage of pore space =  $\frac{W}{Y} \times 100$ .

Porosity is commonly expressed as "ratio of absorption," which is the percentage by weight that the absorbed water bears to the dry

<sup>a</sup> Parks, W. A., Report on the building and ornamental stones of Canada, vol. 1, 1912, p. 61.

weight of the stone. The specimen is dried in a hot-air bath at a temperature of about  $110^{\circ}$  C. until the weight is constant. The stone is then immersed in water for a period of time varying from two days to several weeks. The process of absorption may be assisted by boiling or by placing the immersed block beneath the receiver of an air pump as already described. The difference in weight between the dry and the saturated specimens is the weight of absorbed water. The ratio of absorption of American marbles as determined by the Bureau of Standards varies from 0.0018 to 0.00007.

Pore spaces in marbles permit infiltration of water, which may dissolve the stone, or may cause disintegration by freezing. The evils associated with porosity are discussed more fully under the heading "Imperfections of Marble."

#### STRENGTH.

The strength of a stone is the measure of its capacity to resist stresses of various kinds. It depends partly on the rift of the rock and on the cleavage and hardness of the grains, and partly on the state of aggregation, including degree of cohesion, interlocking of grains, and nature of cementing material, if such is present. Although strength alone is not a sure criterion of durability, a knowledge of the capability of any stone to withstand stresses of various kinds is of great value if the material is to be used for purposes involving extraordinary strains.

Many tests have been made of the strength of building stones. It was early learned from these tests that most stones have many times the strength required for ordinary uses. As pointed out by Buckley,<sup>a</sup> ordinary building stones have 2 to 10 times the crushing strength required in any structure for which they may be used. As a consequence of a recognition of this fact, there was a reaction against making tests, which were regarded as superfluous. An increased demand for strength in structural stone and a wider knowledge of the significance of strain resistance has lately led to a renewed interest in strength tests. It is known that stones are less durable when exposed to intense strains, and it seems reasonable to conclude that the rate of disintegration increases with proportional rapidity as the strain to which the rock is subjected approaches more and more nearly to the ultimate load it is capable of bearing. Rock strength may therefore have a decided influence on the rate of disintegration, even when it is evident that the strength is far in excess of the requirements.

---

<sup>a</sup>Buckley, E. R., *Building and ornamental stones of Wisconsin*; Wis. Geol. and Nat. Hist. Survey Bull. 4, 1898, p. 59.



The tests commonly made are for crushing strength, transverse strength, elasticity, and shearing. As tests can be made only with high-priced specially designed equipment, which is available to few people, a description of the methods of testing is omitted, and the discussion confined to a brief consideration of the significance of the various types of stress resistance of which rocks are capable.

#### CRUSHING STRENGTH.

As already stated, most rocks that, after ordinary superficial inspection, would be chosen for structural purposes have many times the crushing strength required for ordinary uses. For certain purposes, however, such as bridge piers, abutments, columns, and the base blocks of very high structures, crushing strength demands more than ordinary attention. The tendency, more noticeable every year, to increase the height and superincumbent weight of great city structures makes strength tests more and more useful.

Rock structures have a definite influence on strength. As a rule rocks will bear a greater compressive stress across the bedding plane than parallel with the bedding plane. Hence stones should not be laid with the bedding planes vertical.

#### TRANSVERSE STRENGTH.

Transverse strength may be measured by testing the capability of a bar of stone supported at its ends to bear weight exerted at its center. Such tests indicate the suitability of a marble for door or window caps, or as bridging material that must bear a heavy load. Breakage of such caps, however, must not always be attributed to a weakness in the material employed, as unequal settling or improper laying may be the chief causes.

#### ELASTICITY.

When subjected to crushing strain rocks are capable of being appreciably compressed before rupture takes place. A measure of this compressibility in terms of the load is what is known as the modulus of elasticity. Parks<sup>a</sup> defines it more explicitly as follows: "The decrease in length of a bar of material thus subjected to pressure divided into the original length of the bar, and multiplied by the load in pounds per square inch, gives what is known as Young's modulus or the modulus of elasticity or compressibility." Merrill<sup>b</sup> found that after relief from intense pressure below the point of rupture rocks failed to completely recover their original form. This he termed a permanent "set."

<sup>a</sup> Parks, W. A., Report on the building and ornamental stones of Canada, vol. 1, 1912, p. 47.

<sup>b</sup> Merrill, G. P., Stones for building and decoration, 1913, p. 478.



Compressibility may be the cause of cracks in the lower courses of certain large structures. It is evident that building stones having a low modulus of elasticity may under heavy superincumbent load be appreciably compressed with a resultant settling of the structure, and if one part of the building is composed of blocks having a different modulus of elasticity from those of an adjoining part, the settling will not be uniform. Such settling can take place only under extremely heavy loads. A knowledge of the elasticity of marble is, as quoted by Buckley,<sup>a</sup> "valuable in determining the effect of combining masonry and metal or of joining new masonry to old; in calculating the effect of loading a masonry arch; in proportioning abutments and piers of railroad bridges subject to shock, etc."

#### SHEARING.

The tendency to shear—that is, the tendency of one part of a block to slide laterally with respect to another part—is strong in certain structures, such as massive arches and lintels. Certain blocks in large buildings are subjected to strains in different directions, and the tendency to shear may be pronounced. Thus, shearing tests of marble designed for such purposes are of value.

#### WEATHERING OF MARBLE.

The term "weathering" is applied to the disintegration that results from exposure of rock to the various natural agencies that are active at or near the surface of the earth. In previous paragraphs a discussion is given of solubility, porosity, permeability, chemical composition, hardness, texture, and state of aggregation of marble.

The rate of decomposition by weathering is somewhat dependent on these physical features. For example, a soluble rock weathers more rapidly than one that is relatively insoluble, and an open-grained porous rock decomposes more readily than one that is more solid and impervious. Owing to this direct dependence of the rate of weathering on the physical properties of marbles, tests of the various qualities of a given stone are of great value in estimating its probable rate of weathering.

Climatic and atmospheric conditions greatly influence the rate of weathering. Oxygen and carbon dioxide are the most effective atmospheric agents of decomposition. The rate of weathering depends also on temperature. Little or no weathering takes place while the temperature is below 0° C. As the temperature rises, solution takes place with increasing speed and in tropical regions is active throughout the entire year. Rapid changes in temperature

<sup>a</sup> Buckley, E. R., *Building and ornamental stones of Wisconsin: Wis. Geol. and Nat. Hist. Survey Bull. 4, 1898, pp. 63-64.*

cause rapid disintegration, especially in regions where chemical action is supplemented by the effects of frost.

Aside from the effects of frost, changes in temperature produce differential expansion and contraction, which set up shearing stresses, causing flakes to split off. Rupture by expansion and contraction is probably less in the porous than in the nonporous rocks, as in the former case necessary adjustment is made between the grains. As a rule the finer-grained rocks weather less rapidly than do those of coarser grain. Van Hise <sup>a</sup> states that "this is a consequence of the closer interlocking of the mineral particles of the fine-grained rocks, and of the fact that the differential expansion and contraction by change of temperature is less with fine particles than with coarse particles."

Humidity greatly favors decomposition, as chemical action is slow in the absence of water. For example, Cleopatra's Needle, which stood thousands of years in arid Egypt, began to disintegrate so rapidly in Central Park, New York City, that it had to be coated with paraffin to prevent destruction.

The most favorable conditions, therefore, for rapid weathering are coarse and permeable texture of the stone, humidity, warmth, and rapid changes of temperature between points above and below freezing.

Parks <sup>b</sup> conducted a series of interesting experiments designed to test the relative durability of various stones. As carbon dioxide is the most active agent of chemical decomposition, its effect on various stones was taken as an indication of their relative durability. Cubes of stone were immersed in distilled water through which a stream of carbon dioxide was passed, and the corrosive effect was tested by determining the loss in weight by solution in a given time. Those cubes that showed a comparatively small loss by solution were regarded as the most durable.

The change in color of the rock surfaces during the process corresponded closely with the color modification brought about by many years of weathering. Hence such experiments are useful as a means of testing the probable color changes in untried rocks intended for exterior structural work.

Dale <sup>c</sup> investigated the rate of disintegration of marble, especially that used for tombstones. He states that in New England the lettering on tombstones 75 to 100 years old will probably be entirely effaced in 300 years from the date of cutting. He noted a marble slab at Plymouth, Mass., on which the lettering was almost effaced after 87 years of exposure. He mentions also a block of South

<sup>a</sup> Van Hise, C. R., A treatise on metamorphism: U. S. Geol. Surv. Mon. 47, 1904, p. 533.

<sup>b</sup> Parks, W. A., Op. cit., pp. 70-71.

<sup>c</sup> Dale, T. N., The commercial marbles of western Vermont: U. S. Geol. Survey Bull. 521, 1912, p. 38.

Dorset (Vt.) marble on which the edges of the letters are fairly sharp after 79 years' exposure.

Although tests of the various physical properties of marble are useful as an index of its probable durability, actual observations of old structures, outcrops, or quarry walls give the most dependable information. Observations may be made of the rate and uniformity of weathering and the stains or other changes in color involved.

Weathered outcrops give the marble prospector valuable information. On ledges where successive beds are exposed the most enduring beds will stand out in relief. The condition of soundness can also be easily recognized, as weathering emphasizes all cracks or lines of weakness.

Erosion of marble beds commonly results in the formation of "mud pockets," a name given to cavities worn in the rock and later filled with soil. In some places they are large, attaining depths of 50 or 60 feet, and the removal of the clay contained in them is a matter of considerable expense to the quarry operator. In the Tennessee marble deposits erosion has followed the unsoundness and has left numerous masses of rock projecting upward from the solid beds and surrounded by clay. These constitute the so-called "boulder quarries."

Most serpentine marbles are veined with calcite or dolomite, and hence weather unevenly. Moreover, most of them are unsound, and consequently permeable to rain water. Such marbles lose their polish, weather rapidly, become stained, and soon lose both attractiveness and strength if exposed to atmospheric agencies.

#### VARIETIES OF MARBLE.

Marbles may be classed as statuary, decorative, building, and monumental.

Statuary marble is the most valuable variety quarried. It must be pure white in color, uniform and fine-grained in texture, and somewhat translucent, and must possess a marked adaptability for carving. Renwick<sup>a</sup> recommends that "marble for statuary purposes should never be selected in bright weather. Veinings and discolorations are more difficult to discover at this time than at any other. A dull day with a good light is the best time for inspections; if after a shower of rain, so much the better. Provided no rain has fallen, the blocks should be soused with water. Veins and stains can then be more readily perceived. If possible have each block slung and struck with a hammer. If the sound of the blow is dull and heavy, look out for cracks. Should a hard and metallic tone be emitted, the marble will be heavy in working; but if a soft, clear ring is heard, the material is sound and will both work and wear well."

---

<sup>a</sup> Renwick, W. G., *Marble and marble working*, London, 1909, p. 61.



Decorative marble is usually employed in places unexposed to the weather, and hence may be selected for appearance without regard for the effects of weathering. Marble containing large amounts of iron sulphides may therefore be used for interior decoration, and such marbles may give beautiful effects. Brecciated marbles consisting of angular fragments cemented together in nature by calcium carbonate are widely used for columns. Most marbles of this type are imported, only a limited amount having been produced in the United States up to the present. Both pure white and variously colored marbles that possess unusual attractiveness may be used for interior decoration.

Verd antique, or serpentine marble, is in common use. Most of such marble is somewhat unsound and not of sufficient strength to justify its use where it will have to take a heavy load. The marble is commonly used for exterior decoration, but is not to be recommended for this purpose, as it does not weather uniformly and soon loses its polish. For interior decorative purposes it is popular.

Onyx marble possesses a waxlike luster and an attractive banding which make it a popular material for interior decoration.

Numerous statuary and decorative marbles from American quarries are now on the market. No two are alike, and each has its own particular trade name.

Marbles for building and monumental purposes must have attractive and uniform colors, and in addition must possess the ability to withstand weathering and to retain their attractive appearance. For interior decorative marble appearance is the prime factor determining its value, whereas with exterior marble qualities of endurance rank equally in importance with appearance. Building marble should therefore be strong, uniform, close-grained, reasonably nonabsorptive, and free from such impurities as may stain or corrode the surface.

#### DISTRIBUTION AND PRODUCTION OF MARBLE IN THE UNITED STATES.

As stated in the discussion of the origin of marble, the recrystallization of the original limestone is promoted chiefly by heat and pressure. As a consequence most marble beds are confined to areas of extreme folding or igneous intrusions, and hence are to be found chiefly in mountainous regions. In some beds recrystallization takes place without igneous intrusion or movement of the beds, as at Kasota, Minn., but such conditions are rare. The important marble belts of the United States are found in the Appalachian region of the Eastern States and in the Rocky Mountain and the Coast Ranges of the West.



According to figures compiled by Burchard,<sup>a</sup> the States producing marble in 1913, arranged according to value of output, are as follows: Vermont, Tennessee, Georgia, Colorado, Alabama, Massachusetts, New York, Pennsylvania, Alaska, California, Maryland, North Carolina, Utah, Arkansas, New Mexico, Washington, Virginia, and Oregon. Deposits of onyx marbles occur in Arizona, Colorado, Utah, New Mexico, California, Kentucky, and Virginia. Deposits of serpentine marble, or verd antique, occur in Vermont, Pennsylvania, Georgia, Maryland, California, Massachusetts, Connecticut, Delaware, Maine, New York, New Jersey, New Mexico, and northern Michigan.

The value of marble produced in the United States in 1913 was \$7,870,890, and of this amount the three States, Vermont, Tennessee, and Georgia, produced over 76 per cent. The average price per cubic foot was \$2.11.

## THE IMPERFECTIONS OF MARBLE.

### UNSOUNDNESS.

#### MEANING OF UNSOUNDNESS.

When intersected by joints or fissures marble blocks are said to be unsound. The term "unsoundness" refers to all cracks or lines of weakness, other than bedding planes, that cause the marble to break before or during the process of manufacture. The various types of unsoundness are known locally as "joints," "headers," "cutters," "hair lines," "slicks," "seams," "slick seams," "dry seams," or "dries," and "cracks." The term "reed" is applied to a weakness parallel with the bedding.

#### IMPORTANCE OF JOINTS IN MARBLE DEPOSITS.

The presence of joints in marble deposits presents a most serious problem. They may be so close and irregular that the quarrying of profitable material is impossible. Joints should have a marked influence on the mode of quarrying a marble, in order that the waste due to their presence may be reduced to a minimum. The manner in which joints occur and their probable continuance at depth are matters of supreme importance.

#### NATURE OF JOINTS.

Most joints as they appear in marble deposits are straight and uniform though some may be curved or irregular. Some are open and conspicuous and others so obscure that they can be recognized only by those skilled in their detection by long and constant practice.

<sup>a</sup> Burchard, E. F., *Stone: Mineral resources U. S. for 1913*, U. S. Geol. Survey, 1914, p. 1313.

Becker<sup>a</sup> has pointed out that surface tension of water in joints tends to keep them closed. With a space of 0.01 inch the surface tension exerts a force of 13.5 pounds per square foot tending to draw the surfaces together, and if the opening is only 0.001 inch wide the force will be 135 pounds per square foot.

The most striking characteristic of joints is their tendency to occur in parallel systems. The occurrence of two systems approximately at right angles to each other is common. Occasionally a third or fourth system may appear. In exceptional cases joints may present such extreme irregularity that no well-defined system can be recognized.

The spacing of joints is variable. They have a tendency to occur in groups of closely spaced fractures separated by masses in which joints are few in number. In certain Vermont quarries such closely spaced groups are termed "fish-backs." In some deposits joints may be 10 to 30 feet apart, and in others they may be separated by a few inches only. Needless to say, a wide spacing adds greatly to the commercial value of a deposit.

#### ORIGIN OF JOINTS.

In order that one may even approximately understand the distribution, direction, spacing, and persistence at depth of the joints that intersect marble deposits a knowledge of their origin is necessary. Authors generally agree that joints are caused by strains in the rock masses. It is thought that few joints are due to tensile stresses, as joints so caused would show no slickensided surfaces, and be irregular in form, whereas most joints are straight and even planes and are somewhat slickensided. It is now generally accepted that practically all joints are faulted surfaces although the displacement may be small.<sup>b</sup> Daubrée was so firmly convinced of the correctness of this theory that he rejected the term "joint" as failing to imply the existence of relative motion, and introduced the terms "diacalse" and "paracalse." Joints are probably caused by pressure, and pressures in rocks may be highly complex. Curved joints indicate that the direction of effective pressure varies, or varied, from point to point.

The famous experiments of Daubrée<sup>c</sup> indicate that joints may be produced by simple pressure, the joint planes forming at angles of 45° with the line of pressure. His experiments also show that torsion may cause joints. Torsional strain in glass produced two sets of fractures approximately at right angles to one another and usually at angles of about 45° to the axis of torsion. Becker<sup>d</sup> gives

<sup>a</sup> Becker, G. F., The torsional theory of joints: Trans. Am. Inst. Min. Eng., vol. 24, 1894, p. 131.

<sup>b</sup> See Becker, G. F., Loc. cit.

<sup>c</sup> Daubrée, A., Étude synthétiques de géologie expérimentale, Paris, 1878, pp. 300-374.

<sup>d</sup> Becker, G. F., Op. cit., p. 136.

a reason for this arrangement. He shows that torsion of a bar causes diagonal lines to be elongated or contracted, the directions of maximum extension and compression being  $45^\circ$  to the axis of torsion and perpendicular to each other. Cracks in fan-shaped groups may also result from torsion.

Crosby <sup>a</sup> claimed that earthquakes were important factors in the production of joints, each system being parallel with the earth waves producing it. Various systems must therefore have resulted from successive shocks. A second system will usually be nearly at right angles to the first, as an oblique shock would have found relief along the fractures of the first system. The presence of an oblique system presumes a shock of such high velocity that time was not allowed for the strain to find relief along previous fractures. The same author in a later publication <sup>b</sup> shows convincingly how shock and torsion may act in conjunction and produce results according with the phenomena as they occur in nature. His theory, in brief, is as follows:

The torsional theory assumes a very slow, and the earthquake theory a rapid process of joint development. As fractures formed by slow processes are apt to be irregular, following all places of weakness, torsional joints should be irregular. Most joints, however, are fairly regular and are even known to pass directly through the hard pebbles of a conglomerate. Thus, a discrepancy appears in the torsional theory. A single system of regular joints may therefore be ascribed to shock. A subsequent shock in a transverse direction would tend to break up the sheets formed by the joints, the fractures being of a less continuous nature than those produced by the first shock, being terminated abruptly in many places by the first system of joints and continuing in a different though parallel plane. Thus two systems of an unlike character are best explained by the earthquake theory. However, two systems of like character may be referred to the torsional theory except when they exhibit the regularity indicative of instantaneous stresses.

Crosby refers to the idea, upheld by many geologists, of an almost universal condition of strain in the earth's crust, and states that if, while under torsional or folding stresses, the rock is traversed by an earthquake wave, fractures may be precipitated. Experiments with sheets of glass showed that a sudden shock while the torsional strain was distinctly below the breaking point would precipitate the fractures, but that the direction of the fractures was governed by the direction of the axis of torsion. It seems reasonable, therefore, that the result of an earthquake wave traversing rocks under strain would

<sup>a</sup> Crosby, W. O., On the classification and origin of joint structures: Proc. Boston Soc. Nat. Hist., vol. 22, 1882-83, pp. 72-85.

<sup>b</sup> Crosby, W. O., The origin of parallel and intersecting joints: Am. Geol., vol. 12, 1893, pp. 368-375.



be the sudden development of joints, governed in direction by the torsional or folding stresses present.

Becker<sup>a</sup> presents some instructive figures showing the manner in which joints are developed by a compressive force in one direction. In figure 1, A, the force is supposed to act in a direction at right angles to the upper and lower surfaces. Face  $x$  and its opposite face are supported to prevent rupture. The force produces fractures in planes perpendicular to  $x$  and inclined in opposite directions at angles of  $45^\circ$  from the line of force. The face  $x$  is intersected by two systems of joints at right angles to each other. On the other four faces perpendicular to  $x$  they appear as parallel lines but may dip in either of two directions. If considerable deformation takes place before rup-

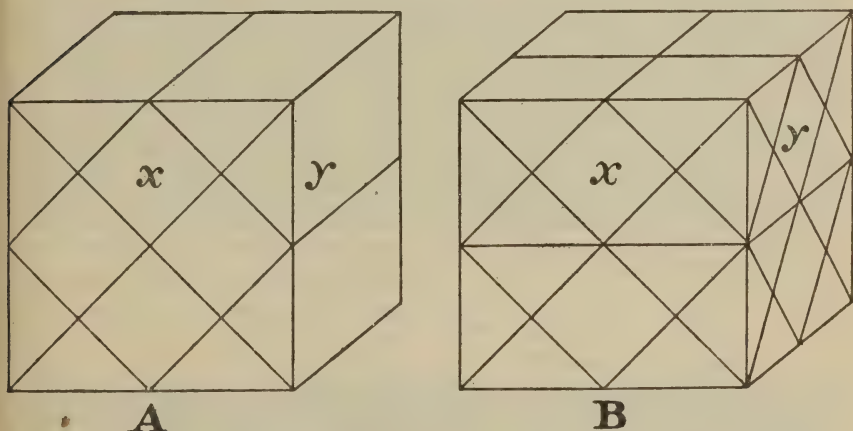


FIGURE 1.—Diagrams illustrating the manner in which a single force may produce several systems of joints—A, joints of two systems, B, joints of four systems.

ture, the joints will make angles of more than  $45^\circ$  with the line of force.

If the support on  $x$  or  $y$  is the same, or, in other words, if the resistance perpendicular to the line of force is uniform, two systems will form simultaneously, as shown in figure 1, B, and horizontal and diagonal cracks will appear on both  $x$  and  $y$ . Thus four systems of joints parallel with octahedral faces will be produced, and the resulting blocks will be octahedral or tetrahedral in form. Other figures given by the author indicate how such systems may be shown to appear on a random plane.

It has been shown in the preceding paragraphs that torsional forces, compressive forces with uniform or nonuniform relief in a transverse direction, or earthquake shocks alone or in conjunction with other forces may produce definite systems of joints. The forces may be

<sup>a</sup> Becker, G. F., Simultaneous joints: Proc. Wash. Acad. Sci., vol. 7, 1905, pp. 267-275.



multiplied and complex, and the resulting joint systems may present a corresponding complexity, but definite systems are generally developed.

The nature of joint surfaces and some inferences therefrom have been discussed by Woodworth.<sup>a</sup>

#### PRACTICAL ILLUSTRATIONS OF JOINT SYSTEMS.

The occurrence of joints as observed in many quarries clearly supports the conclusion deduced by theory in so far as systematic arrangement is concerned. A few

examples selected from many may be offered in support of this conclusion.

In many quarries jointing systems are clear and definite. The direction and spacing of joints as observed at the surface may persist with remarkable uniformity at depths of 100 feet or more. Figure 2 illustrates a system of 21 parallel joints drawn to scale as they appear in an outcrop at a quarry in West Rutland, Vt. They strike N. 82° E. and dip 61° N. 8° W. At a depth of 145 feet they have the same strike, dip, and spacing as at the surface. Four of them are shown in Plate I, A.

Figure 3 shows open joints as they appear in a quarry wall. The horizontal lines represent the various floors cut out by channeling machines. The bedding dips about 40° and is marked by one prominent and several less conspicuous open seams.

Although some irregularities are

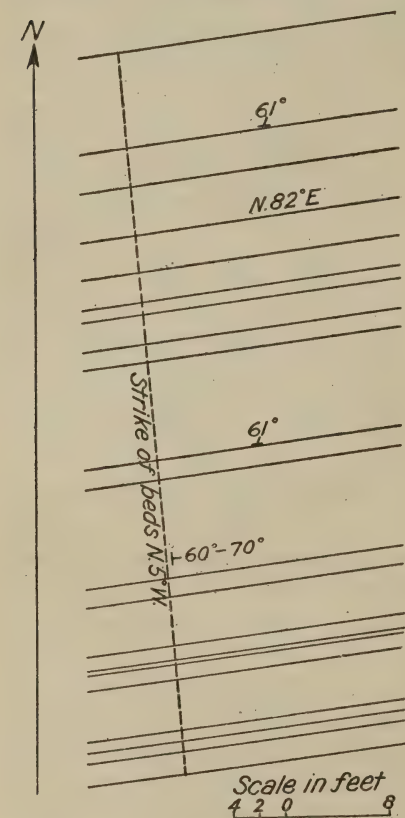
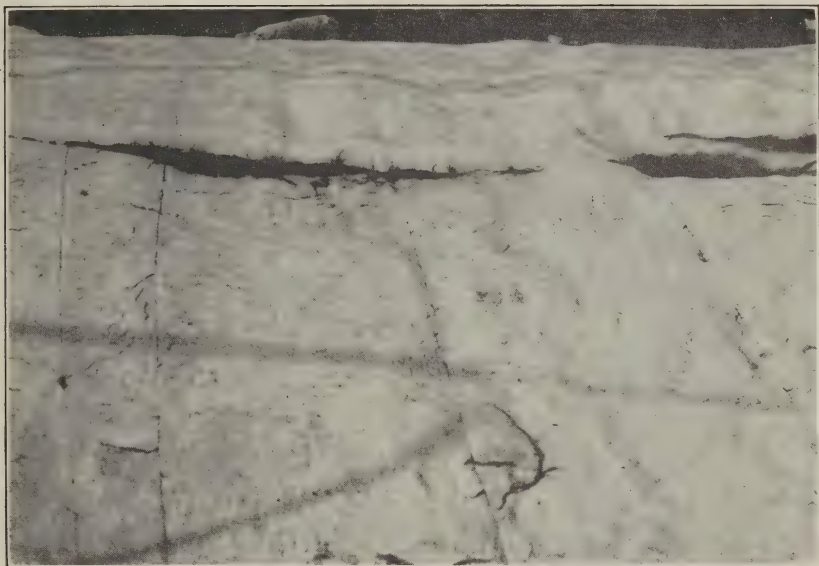


FIGURE 2.—A remarkable system of 21 parallel joints, as found in one outcrop.

to be noted, a definite jointing system may be recognized.

Plate I, B, illustrates a system of parallel joints in a Tennessee marble deposit. Erosion has followed the joint planes and thereby given them exceptional prominence. Attention is directed to figure 11 (p. 67) which shows the joint systems in the floor of a marble quarry in Alabama.

<sup>a</sup> Woodworth, J. B., On the fracture system of joints, with remarks on certain great fractures. Proc. Bos. Soc. Nat. Hist., vol. 27, 1896, pp. 165-183.



A. THE ABRUPT TERMINATION OF JOINTS IN A PLASTIC LAYER IN WHICH DEFORMATION BY FLOWAGE TAKES THE PLACE OF FRACTURES.



B. A SYSTEM OF PARALLEL JOINTS EMPHASIZED BY EROSION.



Hundreds of examples might be given of joint systems, and many quarrymen could supplement them with illustrations from their own experience. On the other hand, in some quarries joint systems may be difficult to recognize. They may be obscure and seemingly rather irregular. However, careful mapping of many of them will reveal definite systems. A practical illustration may be of value.

The operators of a certain quarry declared that the joints in their quarry occurred without any definite system. In order to test the validity of this view the positions of the visible joints were determined as completely as circumstances would permit, with compass, clinometer, and tape measure. A plan, as represented in figure 4, was made showing the arrangement of the joints on the quarry floor. Although some irregularities may be noted, it is evident that only a limited number of joints vary more than a few degrees from a direc-

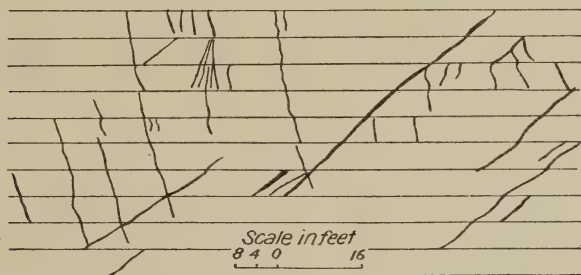


FIGURE 3.—Open joints in a quarry wall.

tion N. 45° W. While variations in dip are rather pronounced, a majority of the fractures are within limits of 10° from the vertical.

It is evident, therefore, that with few exceptions definite systems of joints may be recognized in marble quarries. The economic importance of joints is discussed in detail in a subsequent section devoted to a consideration of channeling in relation to unsoundness.

#### PERSISTENCE OF JOINTS AT DEPTH.

The disappearance or continuance of joints with depth is a matter of profound importance to quarrymen. The belief is common that joints are less numerous at depth. Some persons who hold this opinion can support it by observations in quarries, whereas with others the idea is the expression of an unverified optimism. The origin of joints and the phenomena accompanying their development have a direct bearing on their persistence at depth.

Becker<sup>a</sup> has shown that a mass of rock must occupy a greater space after jointing than before, as cracks and open corners are pro-

<sup>a</sup> Becker, G. F., *Op. cit.*, pp. 274, 275.



duced. Thus as joints demand increased volume in the mass of rock affected, and as surface rocks have freedom of upward motion, whereas deep-seated rocks are more or less restrained in all directions, there is a stronger tendency for joints to form near the surface than at depth. As pointed out by the same author, the pressure on rocks at depth does not obviate the tendency for fractures to form, but may prevent actual ruptures. Deformation without rupture results, as is clearly shown in Plate I, A. The sudden termination of the joints in a plastic layer of soft green marble, as shown near the top of the plate, illustrates the manner in which plasticity permits the necessary adjustment by flowage rather than by fracture.

It is evident, however, that a condition of flowage demands an immense superincumbent load, and takes place, therefore, at depths far

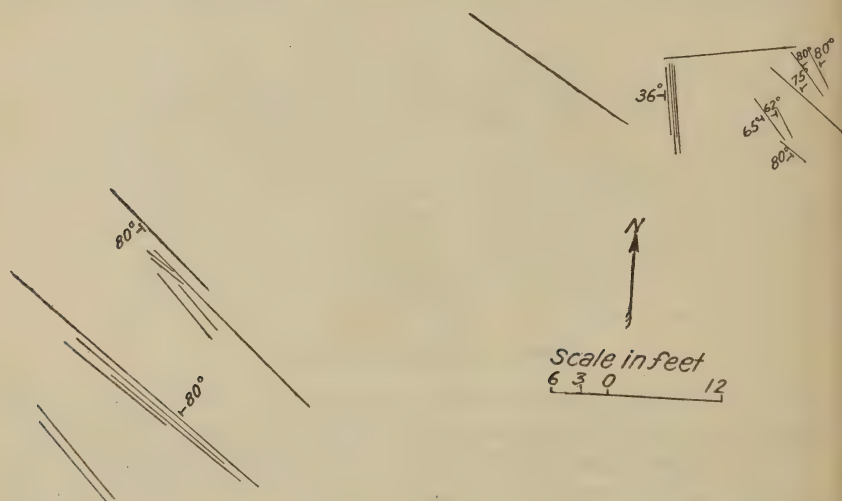


FIGURE 4.—Surface plan of joints that appeared to lack systematic arrangement.

beyond the reach of quarry excavations. If deformation without rupture takes place within the limited depth of a quarry pit, it must be by some other phenomenon than that commonly called flowage.

Deformation may take place at moderate depth by the formation of crushed structures which may be masked by recrystallization. Furthermore, deformation without rupture may take place by twinning, or the development of gliding planes in the calcite crystals. As gliding planes have been produced in the laboratory, it is evident that they may occur in nature under pressures of no greater magnitude than those brought about by a superincumbent load of rock within the vertical range of a quarry pit. Deformation may take place by one of these means at moderate depths.

In theory, therefore, joints are characteristically surface phenomena, and should decrease in number with increasing depth. This theory

is generally substantiated by the conditions disclosed in quarrying. The decrease may not be constant or uniform. Certain beds are more liable to be intersected with joints than other beds, and in consequence one bed may have more joints than those above it. In general it has been found, however, both in quarrying and in projecting core holes, that if any one steeply inclined bed is followed downward the unsoundness constantly decreases.

Certain fractures, known locally as "slicks" or "hair lines," are, as a rule, evenly spaced, vertical, and at right angles to the strike of steeply inclined beds. They usually disappear at depths of 50 to 100 feet. They are regarded as originating from the expansion and contraction of rocks owing to variations in solar heat.

#### UN SOUNDNESS IN VERD ANTIQUE.

Unsoundness in quarries of serpentine marble, or what is commonly called "verd antique," is usually rather pronounced and extremely irregular. It is probably caused chiefly by expansion or swelling, owing to the process of hydration as the serpentine is formed. As a consequence, no definite systems of cracks are to be expected. The formation of lens-shaped masses is common. It is frequently difficult to obtain blocks of any considerable size sufficiently coherent to be of commercial value. Occasionally the cracks are recemented by crystalline calcite, producing an attractive white veining on a green background. The so-called brecciated marbles are extremely unsound masses composed of many irregular and usually angular fragments that have been cemented by chemical precipitation of calcium carbonate.

#### "GLASS SEAMS."

In certain quarries joints that have been recemented in the manner described above are termed "glass seams." They may be sufficiently strong to permit sawing even into thin stock, but most of such seams are planes of weakness. The filling is most commonly of calcite. Occasionally it is of silica, either as quartz, flint, or chert. The silicious filling is least to be desired, as its extreme hardness makes sawing and polishing difficult, and it presents a nonuniform surface. In any case a glass seam usually appears as a conspicuous line which can be regarded only as a blemish when present in otherwise uniform marble.

#### IRON SULPHIDES.

##### THE COMMON IRON SULPHIDES.

The sulphides of iron which occur commonly in nature are pyrrhotite, pyrite, and marcasite. Pyrrhotite is a mineral of variable composition ranging from  $\text{Fe}_5\text{S}_6$  to  $\text{Fe}_{16}\text{S}_{17}$ . It appears to be a solid

solution of  $\text{FeS}$  and sulphur. Pyrite and marcasite are identical in chemical composition, which is expressed by the formula  $\text{FeS}_2$ , but crystallize differently, pyrite being isometric, usually as cubes, octahedrons, or pyritehedrons, and marcasite orthorhombic, with prominent domes and pinacoids. In other respects the two minerals are closely similar and it is difficult to distinguish them.

Pyrrhotite is rare in marble. Pyrite and marcasite are accessory minerals in many marble deposits, the pyrite being the more common. They may appear as scattered crystals of variable size, or may form bands and masses of considerable prominence. The conditions under which pyrite, marcasite, or mixtures of the two may form in nature are discussed by Allen, Crenshaw, and Johnston.<sup>a</sup>

#### MANNER AND EFFECT OF SULPHIDE DECOMPOSITION.

By weathering, pyrite may combine with oxygen to form iron oxide, iron sulphate, or free sulphuric acid. The oxide may stain, and the acid may corrode the rock. When alumina is present whitish crusts of aluminum sulphate and alum may form. As pointed out by Merrill,<sup>b</sup> if magnesia is present the sulphuric acid may produce a soluble efflorescent salt, which forms white patches on the surface.

The process of weathering may, on the other hand, be less detrimental. The pyrite may gradually change color, become coated with a brown crust, and eventually alter entirely into hydrated iron oxide, which retains the original shape of the pyrite and thus forms a pseudomorph.

Marcasite decomposes in much the same manner, though usually more rapidly than pyrite, and more rarely in the pseudomorphic form.

#### SULPHIDES NOT ALWAYS INJURIOUS.

Most authors who have discussed impurities in building stone have stated unreservedly that pyrite is injurious when the stone in which it is contained is used for exterior work. This statement is not always true, however.

Many recorded examples and personal observations show that the rate of decomposition of iron sulphides is different in different deposits. In some marbles the sulphides decompose and form undesirable discolorations in a few months or a few years. Marble from other deposits may contain iron sulphide that will withstand many years of weathering, and show no appreciable change. Examples are known of American marbles containing pyrite that have stood

<sup>a</sup> Allen, E. T., Crenshaw, J. L., and Johnston, John, The mineral sulphides of iron: Amer. Jour. Sci., ser. 4, vol. 33, 1912, pp. 169-217.

<sup>b</sup> Merrill, G. P., Stones for building and decoration, 1903, p. 31.



exposed to the weather for more than 100 years without noticeable staining.

The researches of Julien and Stokes throw considerable light on the conditions governing the rate of decomposition of iron sulphides, and consequently are of great value to quarrymen and architects.

#### RESEARCHES OF JULIEN.

Julien<sup>a</sup> gives a description of pyrrhotite, marcasite, and pyrite with notes thereon that suggest that the process of metamorphism tends to alter pyrrhotite and marcasite into pyrite, and that, of the iron sulphides, pyrite predominates to a greater extent in marble than in the rocks that have undergone less metamorphism.

He points out that pyrite even in the form of crystals is rarely pure, but contains varying quantities of marcasite more or less intimately mixed with it. The author emphasizes the point that the rate of decomposition depends upon the purity of the mineral; mixtures of pyrite and marcasite weathering readily, whereas pure pyrite is stable.

The relation of specific gravity to rate of decomposition is of interest. The specific gravity of marcasite is given as 4.80, and of pyrite as 5.01. As pyrite is the heavier, any admixture of marcasite with it will lower the specific gravity. Consequently specific gravity may be taken as a criterion of purity, and therefore of resistance to weathering.

In judging the probable resistance to weathering of any iron sulphide present in a rock the author recommends careful examination of form, color, fracture, and density with supplementary tests for determining whether the mineral is pure pyrite or whether it contains marcasite mixed with it. If mixtures of the two sulphides are found, rapid alteration is to be expected. If possible, pure material, preferably crystals, should be obtained. The observations to be made and their interpretation are as follows:<sup>b</sup>

1. Crystal form; if isometric the material is probably resistant, though such crystals may contain marcasite.

2. Color; if brass yellow the mineral is probably nearly pure pyrite; if pale yellow or tin white marcasite predominates.

3. Fracture; if conchoidal it is probably pyrite; if uneven or granular, marcasite is abundant or the mineral is otherwise impure.

4. Hardness; the hardness of pyrite is 7 and of marcasite 6.5.

5. Odor and streak; a sulphurous odor and greenish streak indicate the presence of much marcasite.

6. Specific gravity; fairly pure pyrite can be expected only when the specimen has a specific gravity greater than 4.97.

<sup>a</sup> Julien, A. A., On the variation of decomposition in the iron pyrites; its cause and its relation to density: *Ann. New York Acad. Sci.*, vol. 3, 1883-1885, pp. 365-404; vol. 4, 1887-1889, pp. 123-223.

<sup>b</sup> Julien, A. A., *Op. cit.*, pp. 221-223.



7. Observations of weathering effects in quarries or outcrops are recommended as of great practical value.

8. Tests may be made of the comparative rapidity of tarnish when the sulphides are exposed to the fumes of bromine or fuming nitric acid.

#### RESEARCHES OF STOKES.

Stokes <sup>a</sup> verifies Julien in some respects and differs from him in others. He asserts that in general marcasite oxidizes more readily than pyrite, though finely divided or porous specimens of either decompose rapidly. Mixtures of marcasite and pyrite decompose readily, probably owing partly to porosity and partly to electrochemical action between the two.

Attention is directed to the fact that the presence of various impurities, such as quartz or silicates, may lower the specific gravity of pyrite. Density alone therefore is not a sure method of detecting the presence of marcasite in pyrite unless the absence of all other impurities is known.

The author also casts some doubt on Julien's claim that most isometric crystals contain appreciable amounts of marcasite. Although exceptional specimens may contain marcasite, most isometric crystals are pure pyrite, and orthorhombic crystals pure marcasite.

#### METHODS OF DETERMINING STABILITY OF SULPHIDES.

Whether a low density is to be attributed to the presence of marcasite or to some other impurities is of little consequence to the stone man. The important point supported by both authors is that specimens of low density are prone to decomposition. Granular or porous specimens are likewise easily oxidized.

It would seem that by making observations and tests as suggested by Julien a fair conception could be gained of the probable stability of the sulphides present, and therefore of the adaptability of the marble for exterior structural purposes. A microscopic examination is of value as indicating whether the sulphides are present as crystals or in porous or granular form. Crystal form and specific gravity are the physical properties that seemingly give the most definite information. The most reliable information may be obtained from observing structures made of sulphide-bearing marble that have stood for many years exposed to weathering. If such observations can not be made, chemical tests may be substituted. The condition of sulphides in the quarry wall or ledge may give less reliable information. Repeated attacks of circulating ground water contain-

---

<sup>a</sup> Stokes, H. N., On pyrite and marcasite: U. S. Geol. Surv. Bull. 186, 1901, 50 pp.

ing active solvents may decompose the sulphides, although in the dry wall, or exposed only to the action of rain water, they may last indefinitely without sign of decomposition.

#### THE USES OF SULPHIDE-BEARING MARBLES.

Although it is true that iron sulphide is not necessarily injurious in marble, it should be carefully avoided in the selection of stone for exterior uses unless good evidence has been obtained that stains will not result. For interior structural or ornamental purposes, however, sulphide-bearing marbles may be used. In some structures the yellow bands and patches of pyrite have produced beautiful effects in polished surfaces.

#### SILICA.

##### ORIGIN OF SILICA IN MARBLE DEPOSITS.

Silica may be an original constituent of the marble mass. Marble is formed chiefly from an accumulation of calcareous remains, which have been crushed, folded, and recrystallized to a greater or less degree. It is well known that certain marine organisms, such as diatoms and some varieties of sponges, have silicious skeletons. An accumulation of such shells would form masses of silica. The occurrence of flint balls in chalk cliffs is ascribed to this cause. The silica may appear in straight or lenticular bands or knots.

Van Hise<sup>a</sup> claims that as most of the organic silica is in soluble form it is dissolved and later reprecipitated as chert. He emphasizes the strong tendency in minerals to segregate during the process of deposition from solution, and ascribes to this cause the occurrence of the chert in knots and lenses. The larger and more persistent bands may consist of silicious matter in its original form. The occurrence in concentrated masses may be due to the tendency of marine organisms to live in colonies.

Conditions favoring the propagation of silicious-shelled organisms probably prevailed over a wide area at the same time. If therefore silicious lenses are found in a certain bed in a marble quarry, in all probability they are characteristic of that bed over a wide area.

Silica may also be introduced into marble beds at a later stage in the history of the deposit. Water that percolates through fissures in the mass may contain small quantities of silica in solution, and this material may be precipitated in cracks and cavities. Silica in this form will tend to follow the unsoundness, and may even effectually seal up the fractures.

---

<sup>a</sup> Van Hise, C. R., A treatise on metamorphism; U. S. Geol. Surv. Mon. 47, 1904, p. 817.

## DETRIMENTAL EFFECTS OF SILICA IN MARBLE.

Silica is at least twice as hard as ordinary marble, and in consequence its presence greatly retards channeling, drilling, or sawing and is injurious to tools, especially wire saws. A flint ball may divert the saw cut to one side or may impede or entirely prevent cutting. Moreover, the unequal hardness presented by the surface of a flinty marble makes it difficult to obtain uniformity of finish under the buffer.

The presence of silica usually detracts from the appearance of marble. The flinty or cherty mass as a rule differs from the marble in color or texture and thus constitutes a blemish comparable with that produced by the presence of a knot in an otherwise uniform stick of timber.

## SILICATED MARBLES.

Silicated marbles are those that contain silicates such as pyroxenes, amphiboles, mica, or chlorite. Such silicates may result from the combination of silica with the calcium or magnesium of the marble, with the escape of carbon dioxide. Marbles containing interbedded silicates may also be included under silicated marbles, although none of the silicates may have been derived from the marble.

Interbedded silicates may originate from clay beds in the original limestone. Clay brought down by rivers may be interbedded in limestone deposits formed near the shore. The process of metamorphism that changes the limestone into marble has a pronounced effect on the interbedded clay. Ordinarily it is altered into mica and chlorite. The resulting marbles will therefore contain bands of these minerals. In marbles that have undergone a limited amount of folding and deformation the mica and chlorite bands will remain conformable with the original bedding. Such bands may constitute definite bedding planes separating beds of pure marble. In such form they are not serious imperfections and may even assist the process of quarrying. If deformation has been more intense the mica and chlorite may be scattered throughout the marble mass as a dark banding. Although serviceable for certain purposes, in general a clouding or banding detracts considerably from the market value of the stone.

The formation of silicates by combination of silica with the calcium and magnesium of the marble has usually been accomplished by intense metamorphism. Dolomitic marbles may contain crystals of tremolite, a silicate of calcium and magnesium. Most of the crystals are white with a silky luster, have a characteristic diamond-shaped cross section, and are much harder than marble. They may be microscopic in size or may attain a length of 2 inches. They



break out easily and consequently are serious imperfections when of large size. White mica is a common silicate in marble. Its presence tends to make the rock more easily cleavable, as the plates are usually parallel with the grain. Wollastonite, diopside, olivine, and tourmaline are other common silicates occurring in marbles.

#### DOLOMITE IN MARBLE.

A marble that consists of pure dolomite is harder and more resistant to weathering than one consisting of pure calcite. Thus a marble consisting of dolomite may be no less valuable than one consisting of calcite. As there is a marked difference in solubility and hardness between the two minerals, a marble composed of alternating masses of calcite and dolomite is undesirable. If dolomite is present in lenses or bands, unequal weathering will result and produce a non-uniform surface. There is also the probability of differences in texture, color, or susceptibility to polish between the two minerals. Although pure dolomite, or intimate mixtures of dolomite and calcite, are not to be regarded as inferior types of marble, heterogeneous mixtures in the form of lenses, knots, or bands are, for the above reasons, undesirable.

#### UNDESIRABLE COLORS.

Marbles may have various colors or combinations of colors, many of which are attractive for decorative purposes. The colors may be permanent or they may change after exposure to sunlight or weather. The fading of an attractive color is undesirable. Vogt<sup>a</sup> states that certain Norwegian blue marbles fade after five years' exposure to the light. Marbles from various other localities are known to fade perceptibly after exposure to sunlight for a number of years. As marble is usually chosen for its inherent attractiveness, color is one of its chief assets, and if the color changes the rock may no longer be of value. Occasionally exposure to the weather improves the color of a marble, but as a rule permanence of original color is to be desired.

The origin of some undesirable shades of color may be easily determined. Black and gray shades are usually due to the presence of carbon. Red and brown shades may be due to manganese oxide ( $MnO$ ) or to hematite ( $Fe_2O_3$ ). Cream and yellow colors are caused by the presence of fine particles of hydrous iron oxide, limonite ( $2Fe_2O_3 \cdot 3H_2O$ ). If pyrite is present in a marble its appearance may be attractive when the marble is first quarried, but may give rise to stains from subsequent oxidation. Green colors may be due to sericite, chlorite, epidote, actinolite, chrome mica, diopside, or serpentine.

Certain other shades of color are of obscure origin. Some variations in color, though slight, may detract immensely from the market value

---

<sup>a</sup> Vogt, J. H. L., Norsk marmor, Kristiania, 1897, p. 354.



of the rock. A white marble commonly shades off into a lifeless gray. This may be due to imperfect recrystallization of the gray rock, which may therefore be classed as a limestone rather than a marble. In other deposits an attractive white may fade into an inferior bluish white. Lenses and bands of the bluish material may pass irregularly through the white, and thus occasion excessive waste, or necessitate classifying the marble in a lower grade. The reason for this bluish coloration is unknown; some impurity in minute quantities or some peculiarity of crystallization may be the cause.

During the process of limestone formation there is a tendency for similar conditions to prevail over a wide area at the same time. As a consequence colors due to substances that are original constituents of the marble tend to exhibit a minimum variation in different parts of the same bed and a maximum variation in passing from one bed to another. It is usually found that certain beds in a marble deposit give more attractive colors than other beds, and that each bed exhibits more or less constancy of color. The variation from white to bluish white in different beds may be pronounced, though less marked variations have been noted in different parts of the same bed. If marble of a particularly pleasing color appears in a certain bed, there is much greater probability of finding more of the same material by following the original bed than by seeking for it in other strata.

#### FISSILITY.

In certain marble deposits numerous parallel cleavage planes have been developed. They may be so closely spaced that little serviceable marble can be obtained. According to Becker,<sup>a</sup> cleavage in rocks is due to a weakened cohesion along planes of maximum tangential strain (or maximum slide). The process of folding, which is so common in marble beds, undoubtedly causes intense strains, which may develop a cleavage or fissility in the marble. Marbles that are fissile have probably been subjected to profound metamorphism. The same author<sup>b</sup> points out that crystals tend to grow in the direction of least resistance, and hence mica plates, so common in easily cleavable marble, tend to grow parallel with the planes of schistosity. Thus the mica may further facilitate the cleavage. The presence of much muscovite (white mica) parallel with the cleavage planes in several fissile marbles has been noted.

Marble with a certain degree of fissility may be used if cut in slabs parallel with the cleavage. There is a probability of great waste, however, and as a rule such deposits should be avoided.

---

<sup>a</sup> Becker, G. F., Experiments on schistosity and slaty cleavage: U. S. Geol. Survey Bull. 241, 1904, p. 11.

<sup>b</sup> Becker, G. F., Op. cit., p. 22.

**DEFECTS IN TEXTURE AND STATE OF AGGREGATION.**

Uniformity of texture is an important requisite of a marble. A noticeable variation in the size of grain from point to point detracts greatly from its appearance. Moreover, different textures decompose at different rates, and therefore nonuniform marble will tend to weather and decay in an unequal manner, and to produce a pitted and uneven surface.

Marbles vary greatly in size of grain. Most of the fine-grained marbles are more durable than the coarse grained. As pointed out by Dale,<sup>a</sup> acid water travels more rapidly between coarse than between fine grains. For all exterior structural purposes, therefore, fine-grained marble is to be preferred.

Aside from its durability, the fine-grained marble offers other advantages. It usually takes a better polish than coarse-grained material. Moreover it is as a rule better adapted for intricate cutting or carving, though some of the coarse-grained marbles of Georgia carve well.

The percentage of pore space is variable in marbles from different localities. In no marbles are the crystals so firmly cemented together that no pore space exists, though in some specimens it is small. Pores permit the infiltration of water, which affects the marble in two ways. Rain water may contain carbon dioxide or other solvents that hasten the decay of the rock. Water in the pores may freeze in cold climates and thus cause disintegration. Hence, low porosity is desirable in marbles for exterior uses.

However, a marble should not be condemned simply on the basis of percentage of pore space. A rock of low porosity may under certain conditions decompose more readily than one of much higher porosity but having a different type of pores, as explained in the following paragraph:

The rate of solution of marble by circulating water depends on the nature of the solution and the rate of circulation. The nature of the solution depends on environment, and not on any inherent peculiarities of the marble itself, except in so far as the solution is modified by substances dissolved during its passage through the rock. The rate of circulation, however, depends largely on the nature of the pores. If the pores are isolated from each other by walls of rock, circulation must be slow. If, on the other hand, they are connected by open channels, free and rapid circulation is possible. The rate of circulation depends, therefore, rather on permeability than on percentage of porosity. The permeability of marble may be tested by investigating the rate of flow of water forced through it under pres-

<sup>a</sup> Dale, T. N., The commercial marbles of western Vermont: U. S. Geol. Surv. Bull. 521, 1912, p. 37.

sure as described by Parks,<sup>a</sup> or it may be tested by the distance to which colored aniline dyes will pass into the rock in a given time. Naturally those marbles of high permeability are undesirable for exterior work.

In cold climates the effect of frost on exposed marble may materially increase the rate of decay. Although low porosity undoubtedly tends to diminish the danger of deterioration through frost action, the effects of frost are not proportional to the percentage of pore space. As shown by Buckley,<sup>b</sup> the effect of frost on a stone depends on the rapidity with which the stone gives up its included water. Parks<sup>c</sup> in commenting on this statement brings out the following facts: The injury is caused by the expansion resulting when water changes into ice. If the pores are full of water, expansion must cause disintegration of the rock, but if the pores are only partly full, expansion may take place without rupture of the pore walls. Capillarity may keep many of the finer pores full of water, whereas air in the larger cavities prevents complete filling. It is evident, therefore, that a stone having a large proportion of fine pores is in greater danger from frost than one in which the proportion of fine pores is lower. In cold climates, therefore, numerous fine pores are undesirable in marble that is to be exposed to the weather.

Porous stones readily collect dust particles and therefore become dirty much more rapidly than compact varieties. Porous stone should therefore never be used if it is to be exposed to excessive smoke or dust.

In sawing porous marbles sand must be used as abrasive. Although crushed steel cuts more rapidly, it may enter the pores, and later scratch the slab during the process of polishing, or may by rusting cause serious stains.

The strength of a marble depends in part on the state of aggregation of its constituent particles. Marbles in which the particles adhere to one another with great tenacity are stronger and more durable under pressure than those in which the grains are loosely coherent. A remarkable interlocking of crystals has been noted in some marbles, and it is thought that such a condition increases their crushing strength.

---

<sup>a</sup> Parks, W. A., Report on the building and ornamental stones of Canada: Can. Dept. Mines, vol. 1, 1912, p. 61.

<sup>b</sup> Buckley, E. R., Building and ornamental stones of Wisconsin: Wis. Surv. Bull., 4, 1898, p. 22.

<sup>c</sup> Parks, W. A., Op. cit., pp. 63, 64.



**PROSPECTING AND DEVELOPING MARBLE DEPOSITS.****PROSPECTING.****VALUE OF GEOLOGIC MAPS.**

Most marble beds outcrop in long and narrow bands, which may extend for many miles. These bands represent truncated edges of folds in the rock and may be curved or straight, depending upon the topography and on the nature of the fold. Much of the rock surface may be covered with gravel, sand, or clay to a considerable depth. The geologist may, by a careful study of outcrops exposed here and there, obtain a knowledge of the chief structural features, and may thus determine the position, thickness, and attitude of the marble beds with a fair degree of accuracy, even if they are entirely hidden by surface débris. If geologic maps of marble belts are carefully made they are of inestimable value to the marble prospector. By accurately locating himself in the field and carefully consulting a geologic map the prospector may determine the position of marble belts beneath the soil and know something of their extent and attitude although they are unseen. It is important, therefore, that all available geologic maps of the region be consulted freely.

**NEED OF DETAILED PROSPECTING.**

Knowledge of the suitability of any particular site can be gained only by detailed prospecting including a determination of the depth of overburden and of surface alteration of the rock and of the extent, quality, impurities, and soundness of the deposit. It is extremely unwise to proceed with development work without a reasonable assurance that an available mass of sound and attractive marble is sufficiently uniform in quality and abundant in supply for profitable exploitation.

**DETERMINATION OF OVERBURDEN.**

The depth of stripping necessary may be determined at small cost by putting down drill holes. The need of such preliminary tests should be recognized. In certain instances stripping has been attempted without any previous investigation of the depth of soil to be removed. The great loss that may result from thus working blindly may be illustrated by one particular instance. A pit opened by a certain quarry company and later abandoned is 60 by 80 feet in extent and 20 feet deep. To make such a pit required the removal of approximately 3,550 cubic yards of soil, which, at an average cost of 25 cents per cubic yard, would have cost \$887. The only purpose this pit actually served was to show the owners that the stripping was too deep to make quarrying at this point profitable. The same informa-



tion could have been obtained by projecting two drill holes each 25 feet deep. At the ordinary cost of rock drilling, \$2 per foot, the cost would have been only \$100, a saving of \$787.

In estimating the necessary cost of stripping for a new quarry the attitude of the marble beds must be taken into account. If the beds are flat a greater area of rock must be uncovered than if they are steeply inclined or vertical.

Naturally conditions relating to disposal of stripping are of great importance. In certain places it is a matter of some difficulty to find a suitable place in which to deposit the soil that must be removed; in other places the soil may be carried to neighboring valleys or low-lying areas.

#### SURFACE STUDY INSUFFICIENT.

Surface observations are of great value, especially as regards jointing. The process of weathering tends to emphasize all unsoundness and thus facilitates the study of joint systems. Exposed surfaces may also permit a determination of dip and strike and the thickness of the beds.

In determining the quality of a marble deposit a study of uncovered knobs or ledges should not, however, be deemed sufficient. On account of surface weathering the top rock may differ materially from the deeper parts of the deposit. Moreover, the number and spacing of joints at the surface may be no indication of the prevailing conditions at depth. Before the prospective quarry operator installs expensive derricks and hoisting machinery and purchases channeling machines and drills he should have a fair idea of the quality and soundness of the marble and the supply available. In order to obtain this knowledge drill cores should be taken at several points.

#### DIAMOND-DRILL PROSPECTING.

The ordinary diamond drill will give the necessary information regarding color, uniformity, and general appearance of the stone, and also the extent of the formation. It will not, however, give definite information concerning the dip and the strike or the unsoundness of the marble. If drill cores come out in long, unbroken sections which show no indication of cracks, it may be assumed that the rock is fairly sound. If, on the other hand, the core is in short sections, the rotation of the drill will in most cases have so worn and rounded the broken ends that it will be impossible to determine whether the breaks are due to natural planes of weakness in the rock or to the process of drilling itself.

## MECHANISM OF THE DOUBLE CORE-BARREL DRILL.

A method of prospect drilling that has been employed by at least one operator involves the use of the double core-barrel drill that was designed primarily for drilling bituminous coal, and operates in such a manner as to bring out a core from a delicate material with a minimum of breaking or other damage. The drill, shown in Plate II, consists of an outer and an inner tube. The outer is attached to the drill rod by means of the core-barrel plug, and is rotated the same as the cutting tools. The inner tube is suspended by a ball-bearing plug at the top and centered by a ball bearing near the bottom. Thus, while the outer tube revolves, the inner tube which carries the core ring or lifter at its lower end remains stationary. The core passes up through the bit into the inner barrel, where it is protected from friction from the rotating parts, and from the washing action of the water. An ample water supply is provided between the two tubes. The purpose of the inner tube is to hold the core as immovable as possible, and thus to prevent the wearing or rubbing of the broken ends. Although the core does not revolve after it passes through the core lifter into the inner tube, the principle of cutting and the action of the bit on the core until after it passes into the inner tube are just the same as with the single-barrel drill. Therefore, if a natural parting is encountered in the rock, or if for any reason the core is broken off, it is liable to be rotated somewhat before it gets into the inner tube. If the core breaks obliquely the danger of rotation is minimized. Cores should be at least  $2\frac{1}{2}$  inches in diameter.

## VALUE OF DOUBLE CORE-BARREL DRILL TO PROSPECTOR.

The use of a drill like that described enables the prospector to judge the unsoundness of the marble at points beneath the surface. If one examines carefully the ends of the sections of drill core one can, almost without exception, interpret the breaks, and state whether they are due to natural joint planes in the rock or to the process of drilling. If the cores are properly oriented, the proximity and direction of all natural cracks in the rock in the immediate vicinity of the drill holes may thus be ascertained.

If the marble deposit is well exposed, the dip and the strike may be determined from examination of the outcrops. If, however, it is completely buried, these features may be determined from the drill cores if they are carefully oriented.

## DRILLING THE HOLES.

## NUMEROUS HOLES UNDESIRABLE.

Information should be obtained with a minimum number of drill holes. In this respect prospecting for marble differs materially from prospecting for metalliferous ores. As regards metalliferous ore, the soundness of the ore is not important, whereas with the marble every crack or cavity increases the proportion of waste in the quarried product. A drill hole in a quarry is quite as objectionable as a crack. If the deposit lies flat or nearly so, a single well-placed core driven entirely through the deposit will give information as to the character of the marble, and show whether it is one homogeneous mass, or is divided by streaks of color or open beds into different layers, and whether the layers differ in character one from the other.

## PROSPECTING THICK BEDS DIPPING AT A MODERATE ANGLE.

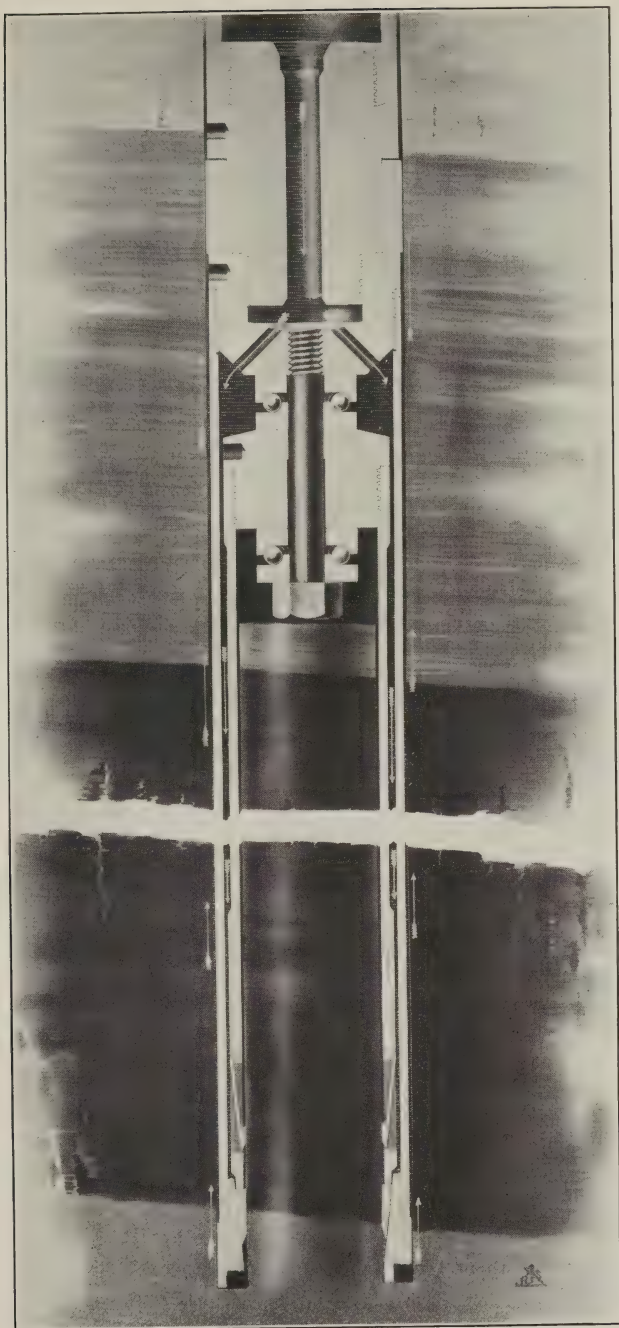
If the deposit dips at a moderate angle and is comparatively thick, the best way to determine its thickness and the character of its beds is to lay out a line of drill holes at right angles to the strike. The first drill hole that penetrates the upper beds should begin in the hanging wall, a name given to the bed immediately overlying the marble band. The holes should be of such depth and spacing that the bottom of a hole in the upper beds will penetrate the same layer as the top of the neighboring hole on the side toward the footwall. The core nearest the footwall should reach and penetrate this wall. By this method a series of core holes of moderate depth will supply samples from all the beds, and the relatively high cost of drilling deep holes will be avoided. The same information could be obtained by drilling one hole, starting at the hanging wall and penetrating the entire deposit to the footwall, but the cost would probably be much higher.

## DETERMINATION OF DIP AND STRIKE FROM DRILL CORES.

If the marble has at any time been subjected to severe stresses, and has been sufficiently plastic to flow freely under the pressure, the original distribution of color may be so disturbed that it no longer bears any well-defined relation to the bedding. Ordinarily, however, the direction of veining or clouding is approximately parallel with the bedding. A vertical core may therefore enable the prospector to determine the average angle of dip of the marble if he measures the angle at which the streaks of color or the grain of the rock traverses the core itself.

If access to the surface of the marble can be obtained before the core is started, the direction of strike may also be determined from a single vertical core by orienting the upper part of the core as it was





DOUBLE CORE-BARREL DRILL.



before drilling, and then by carefully matching the successive pieces. If the surface is inaccessible three cores distributed on the corners of a triangle may serve to determine both dip and strike, provided there are well-defined beds that may be matched in the different cores. If a sufficient area of the marble is stripped the strike and dip can be determined by direct observation, but even then the indications of the core may be of value, because the strike and the dip may vary considerably even within short distances.

#### ARRANGEMENT OF HOLES TO TEST VARIATION IN QUALITY.

It may seem desirable to test the uniformity in quality of marble in the same bed at different points along the strike. To get a fair idea of any variation in quality it is well to lay out, across the strike, two parallel lines of holes, 300 to 1,000 feet apart, depending on conditions. Every effort should be made to locate the two lines of holes so that each core in the second line will come from the same part of the deposit as the corresponding core of the first line. Thus, fair conclusions can be drawn as to the variation in quality of any layer of the deposit as a whole from point to point. All drill cores should be polished on one side in order to facilitate determination of color, uniformity, and degree of polish that may be obtained.

As core drilling is rather costly, it is well to supplement the evidence of the cores by stripping the marble along each line of holes. It is wise also to dig an occasional trench at right angles to each line of core holes so as to expose the marble to some extent along the strike.

#### ARRANGEMENT OF HOLES TO DETERMINE UNSOUNDNESS.

A marble deposit in which the color, texture, or other qualities are highly satisfactory may nevertheless not warrant commercial development because of joints. Most joints occur in two systems, the cracks in each system being approximately parallel with each other, and the two systems more or less at right angles. Occasionally more than two systems are present. The spacing of the cracks is subject to wide variation in different deposits and even in different parts of the same deposit. Near the surface the cracks are usually more numerous and more irregular than at depth. Nearly always at least two systems of cracks will persist with more or less prominence to almost any depth to which quarrying operations may be economically carried. It is important to determine as early as possible which of the cracks that appear at the surface are likely to persist and also their nature and spacing in the deeper parts of the deposit.

Most of the cracks are nearly vertical and also nearly at right angles to the bedding of the marble, few cracks deviating from a



right angle more than  $30^{\circ}$ . Hence, a vertical core taken out of marble that is rather unsound may reveal the presence of few of the cracks. Therefore, a vertical hole is not reliable as a means of estimating the unsoundness to be encountered.

Having determined by any means available the systems of cracks that will probably persist at depth, the general compass bearing of each system should be determined. A series of core holes approximately parallel with the bedding should be laid out, the compass bearing of the core hole being such that it will intersect both systems of cracks at approximately the same angle. A core projected in such a direction will give a fair estimate of the unsoundness present. It also results in an oblique break in the core wherever it crosses a crack of either system, and thus makes less likely any grinding of the ends of the core sections. The origin of a fracture may be determined with reasonable certainty, because a freshly broken surface made in sound marble differs materially in its appearance from the surface of a break due to a joint.

It is important to take cores from near the top, from near the middle, and from near the bottom of the deposit, because the unsoundness often varies in different beds as well as in different parts of the same bed.

If the marble deposit stands at a high angle, one set of core holes driven in an inclined direction and penetrating from the hanging wall to the footwall, or the reverse, can be laid out so as to give the information required as to the quality of the stone and also the unsoundness.

It is practically impossible to take out good cores that are representative of the deposit from horizontal drill holes. The core from a horizontal hole invariably breaks into short pieces, which grind on each other, in spite of the use of the double-core barrel. The core hole has to be slightly larger than the outer diameter of the core barrel in order to permit the passage of water and slush and also to prevent binding of the drill in the hole. As a result the progress of the drill brings a considerable weight on the unbroken end of the core, and as soon as the core is a few inches long it breaks off. The break occurs seemingly at the bottom of the hole, the result being that the spring clips on the inner core barrel, which are supposed to hold the unbroken end of the core and thereby prevent rotation of the inner core barrel, have nothing on which to hold. During the time that the drill is cutting a little farther in and the clips are getting a fresh hold, the inner core barrel rotates to a greater or less extent, so that the core ends grind on each other. Therefore, if the marble beds lie flat or nearly so, unsoundness must be prospected for by inclined core holes; otherwise the cores will not yield the information desired.

## ARRANGEMENT OF THE CORES OBTAINED.

In order to get the fullest information from an inclined core hole the core parts should be matched up from one end to the other, and placed, a part at a time, on an inclined rack that will hold the core in a position parallel with the hole from which it was taken. While the core is in this position the compass bearing of the cracks and also the angle that they make with the core can easily be determined. From this information a plan may be made from which the probable percentage of marble unaffected by unsoundness may be computed with reasonable accuracy.

Before an inclined core is started, the first piece should be marked so that it can be laid top side up after it has been removed. The dip of the core hole should be slightly different from the dip of the beds so that a streak of color, the grain of the marble, or any other feature parallel with the bedding will traverse the core at a slightly oblique angle. With such a method of drilling, if clean right-angled breaks occur, one piece of core may easily be matched to the preceding piece, and the prospector may be sure that he has the entire core properly matched from one end to the other.

## VALUE OF THE INFORMATION GAINED FROM CORES.

From a careful study of cores taken out as described above, valuable information can be obtained in reference to the unsoundness existing at some depth in the deposit. It will often happen that, although the marble appears so broken at the surface that profitable operation seems impossible, investigation with a drill in the manner above described will reveal the fact that the cracks close and disappear to such an extent that profitable operation may be possible after the top rock has been removed.

A later section discusses the advantages of quarrying in accordance with unsoundness, and shows that a knowledge of the extent, direction, and character of the unsoundness is important in laying out the preliminary opening in order to avoid unnecessary expense.

## COST OF MARBLE PROSPECTING.

Prospecting as described above is rather costly. The average cost of double core-barrel drilling by contract is about \$3 per lineal foot. If a marble company has sufficient work to justify owning its own drills the cost can be considerably reduced. In any case the information desired can not be obtained for less than \$5,000 to \$20,000. Many marble deposits, however, can not be profitably worked except on a large scale, so that a considerable outlay made to determine whether conditions are favorable is money well expended.

## PRESERVATION OF CORES.

As a rule, drill cores are not preserved with sufficient care by quarrymen. They are often carelessly stored, lost, or given away as samples. It is important that every part of every drill core be carefully marked and stored for future reference. It must not be assumed that the value of drill cores disappears after their first investigation. They are invaluable records which should be available at all times.

## STRIPPING.

## ORDINARY METHODS OF STRIPPING.

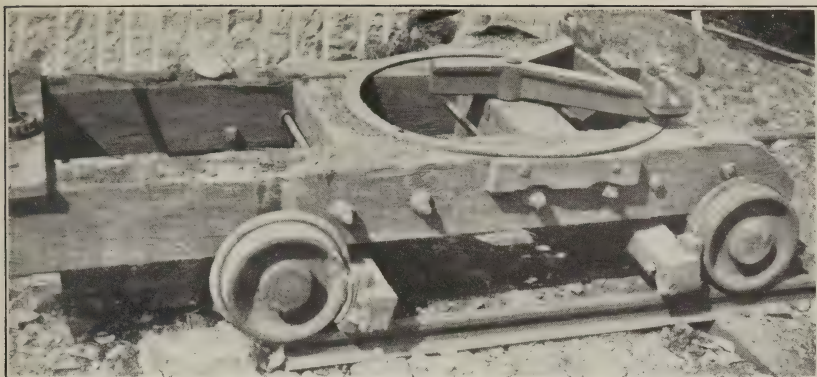
Most marble quarried is of a high grade, and the bulk of rock removed is small in comparison with the quantity handled in many quarries where rock is obtained for cement or road construction. As a consequence the area stripped is usually not great enough to justify the use of steam shovels.

When the surface of a deposit is fairly level, teams and scrapers may be used to advantage. Usually, however, the overburden of soil is so placed that it must be removed by means of large pans which are loaded by hand, and handled with a derrick hoist. In order to remove the soil to a sufficient distance from the excavation, cars and tracks may be necessary. A serviceable car for this purpose is shown in Plate III, A. The loaded pan is placed on the car by means of the quarry derrick. Horizontal iron bars on the undersurface of the pan are placed in the iron sockets of the car. These trunnions are so placed that a little more than half the weight of the load is toward the rear end of the car. When the car reaches its destination, the back of the pan is raised, thus overbalancing the load, and dumping it from the front of the car. Where soil must be removed to a great distance, the loaded pan may be dumped into railroad or cable cars.

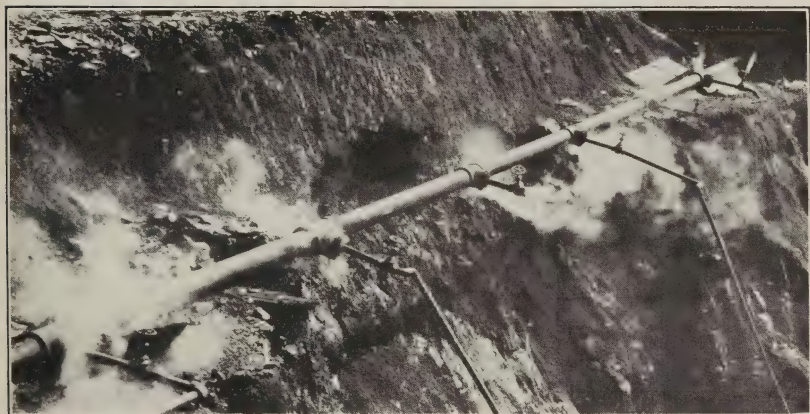
## INSUFFICIENT REMOVAL OF STRIPPING.

A common mistake in the process of stripping is to remove the soil and waste to an insufficient distance from the excavation. The desire to attain quick results at small expense, and lack of foresight regarding the probable extent of future operations are the chief causes of insufficient removal. As a consequence, quarrymen may find after a few years' operation that they must handle material a second time, thus adding greatly to the expense of quarrying. The extensive marble workings of Italy, in the neighborhood of Carrara, Massa, and Seravezza, are greatly hampered by accumulations of rubbish which have buried vast quantities of good marble. The market price is too low to allow the removal of this débris.

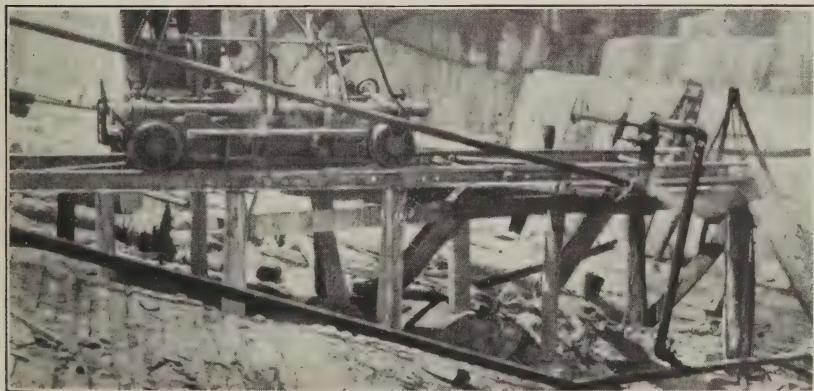




A. SERVICEABLE CAR FOR REMOVAL OF WASTE AND STRIPPING.



B. TYPICAL EXAMPLE OF STEAM LEAKAGE IN A QUARRY TRANSMISSION LINE.



C. TRACK SUPPORTED IN LEVEL POSITION FOR "SAW-TOOTH" FLOOR CHANNELING.





Lack of foresight is also shown at some quarries where quarrymen dump their waste material into abandoned excavations. This procedure may be justifiable if there is no probability that the pit may ever be reopened. The practice has been observed, however, in places where it seemed that greater success would have been attained by going deeper in the old pits than by making new ones.

#### WASTE HEAPS IMPEDING FUTURE DEVELOPMENT.

Another mistake in the disposition of waste material is due to the inability of the operator to foresee the direction in which future operations will extend his quarry. As regards steeply tilted beds, it is obvious that if excessive depth is to be avoided development can take place only in the direction of strike. Nevertheless, at certain quarries of this type observed, stripping has been deposited directly in line with the strike of the beds, and thus an extension of the quarry excavation must soon overtake a great heap of accumulated débris.

#### USE OF OVERHEAD CABLEWAY HOISTS.

For the removal of both soil and waste rock to a sufficient distance to avoid interference with future operations an overhead cableway hoist may be desirable. At one quarry at which such a cableway is used the hoist engine is mounted on a truck that travels on a curved track. It can thus be shifted to strip in different places. Where difficulties are in the way of removal of stripping by derrick or car, the advisability of adopting a cableway system may be considered.

#### HYDRAULIC STRIPPING.

Hydraulic stripping is employed with success in several marble quarries. There are certain conditions, however, that must be met in order that hydraulic stripping may be successful or even possible. Two important conditions are an adequate water supply and easy drainage. If the water supply is obtained from drilled wells or small streams that may go dry, the process will probably fail.

The soil removed by hydraulic stripping may be disposed of in several ways. It may be carried away in a stream valley and deposited naturally at various points along the course of the stream. In some places, however, the deposition of the soil along the stream valley would be detrimental to agriculture or to other interests. In that event, a dam may be built forming a settling basin. In certain operations, surface soil is conveyed to abandoned quarry pits. With such disposal surface drainage is also necessary in order to get rid of accumulated water. Also one must be sure that there will be no future desirability of reopening the pit, as reopening would not be feasible after the pit was once filled with soil.

In certain quarries in Georgia, the good marble is underlaid by a hornblende rock. When in the process of quarrying the hornblende rock is reached, the pit is abandoned, and can then be used as a settling basin in the process of hydraulic stripping.

Those who have had experience in hydraulic stripping estimate that the cost of soil removal by that method may be as low as 2 cents per cubic yard.

### GENERAL PLAN OF QUARRYING.

#### PRODUCTION OF UNIFORM GRADES OF MARBLE DESIRABLE.

Success in marble quarrying necessitates an adequate supply of uniform material. The inherent qualities of certain marbles win immediate popularity. With other types it has been found that by profusely scattering samples and following them with structures in various localities, public taste may be educated to demand a certain type of marble. It is most discouraging for a quarryman who has won popular favor to find that he can no longer match his samples because of failure of the deposit. The desirable bed may have pinched out, or insufficient production may be due to the operator's inability to foresee the general plan of quarry development that ought to have been followed. If the desirable bed lies flat and near the surface, extensive stripping is necessary. If the bed is inclined and the overlying beds are too unsound to permit tunneling, quarrying must follow along the strike of the beds, also demanding extensive stripping. If in either instance the cost of stripping is excessive, the quarrying of the bed must be abandoned.

#### FACTORS GOVERNING PLAN OF DEVELOPMENT.

The most successful mode of operation can not always be foreseen. Certain geologic factors, however, may give information pointing to the most logical method of developing the deposit. Three chief factors are the attitude of the beds, the depth of overburden, and the uniformity of the product in a given bed and in successive beds, and these three factors are intimately related. Before operations are begun the intelligent prospector will determine, in a general way at least, the necessary depth of stripping, the dip, and the strike of the beds, and the uniformity of the marble, so that he may plan a logical quarry development. The influence of the various factors on the plan of operation is discussed in the following paragraphs.

#### EFFECT OF ATTITUDE OF BEDS ON PLAN OF QUARRYING.

Most marble beds are situated in regions of extreme folding, and most of the beds, although originally flat, are tilted, though a few are level or nearly so. The attitude of the beds is of great importance to



the quarryman, especially if the various beds differ from each other in texture or color. If beds are inclined at a moderate angle, either long shallow quarries or tunnels must be made in order to keep up the supply of stone from a particular bed. If the beds are steeply inclined or vertical, either long, shallow quarries involving a great area of stripping, or deep quarries with their associated dangers and expense are necessary. If the strata are flat and the desirable bed is near the surface, a wide, shallow quarry results.

It is clear that if beds lack in uniformity, their attitude has a direct bearing on the most desirable plan of quarrying. If, however, the bedding is indistinct and a number of contiguous beds are uniform in color and texture, as in certain quarries in the Knoxville region of Tennessee, the attitude of the beds has a minor influence on the type of quarrying to be employed.

#### EFFECT OF OVERBURDEN ON PLAN OF QUARRYING.

If the marble lies in approximately horizontal beds of limited thickness, the production of large quantities will necessitate the stripping of a wide area. If the overburden of soil or superficial waste rock is great, the cost of stripping may absorb all profit. If a sufficiently strong roof is available, extensive stripping may be avoided by employing tunnel methods.

As regards flat-lying, uniform beds of great thickness, a heavy overburden will tend to promote deep quarrying, whereas a light overburden will encourage the development of wider and shallower pits.

If the beds are vertical or steeply inclined, a heavy overburden will result in deep quarrying or tunneling, whereas if only light stripping is necessary a greater lateral development in the direction of the strike is possible.

#### EFFECT OF UNIFORMITY OF PRODUCT ON PLAN OF QUARRYING.

In a few deposits thick beds are uniform throughout. However, in many marble regions a certain bed supplies stone of better quality than the beds above or below it. Obviously it is desirable to develop the quarry in such a manner as to obtain a maximum supply of the high-grade material. The shape of the opening, whether it shall be deep or shallow, open pit or tunnel, depends chiefly on the attitude of the beds and the depth of stripping required. Thus one may see how intimately the three factors of attitude, overburden, and uniformity are related, and how necessary it is that they be understood before development is attempted.

**QUARRY OPERATIONS AND EQUIPMENT.****POWER PLANTS.****TYPES EMPLOYED.**

For a discussion of various power plants and their relative advantages and economies the reader is referred to the work of Brunton and Davis.<sup>a</sup> On account of the special types of machinery employed in marble quarrying, some additional notes bearing particularly on this subject are given herein.

The following types of power plants have been observed in marble quarries in the United States: (1) Steam only; (2) steam for channelers and tripod drills, with auxiliary air compressor for hand drills; (3) compressed air generated by steam and conducted to the quarry by pipe line; (4) electricity developed by steam; (5) compressed air developed by purchased electric power and transmitted to the quarry by pipe line; (6) purchased electricity transmitted directly to the quarry; (7) hydroelectric power plants owned and operated by the quarry company, the electricity being transmitted directly to quarry machinery.

**ADVANTAGES OF ELECTRICITY.**

The superiority of electricity over any other form of power for quarry operation has lately been demonstrated. This superiority is due chiefly to the recent development of electrically operated machinery. The electric air channeler which is now in common use consumes on an average only 10 horsepower for the machine itself, although a 15-horsepower motor is required to drive it. A steam or air channeler requires a steam capacity in the boiler that, if applied to an economical generating unit, would develop at least 50 to 60 horsepower. The difference in power consumption between the electrical and other machines is due partly to differences in transmission or transformation losses, and partly to the higher efficiency of electrical machinery generally. The electric air channeler is, therefore, much more economical of power for the same results than the other types.

If the nature of the rock is such that "jackhammers"<sup>b</sup> are used in lieu of other types for quarry drilling, an air compressor is necessary. Should a direct-acting electric drill be produced that can do the work of a jackhammer, compressed-air transmission lines might be eliminated from the quarry. Such a drill has not as yet been developed, at least not in a practicable form.

<sup>a</sup> Brunton, D. W., and Davis, J. A., Safety and efficiency in mine tunneling: Bull. 57, Bureau of Mines, 1914, 271 pp., 6 pls., 45 figs.

<sup>b</sup> Term applied by the manufacturer to a nonreciprocating rock drill, worked without a tripod, and provided with an automatic rotating device. The word is also spelled "jackhammer" in mining literature. It uses hollow steel through which the exhaust air passes and blows the cuttings from the drill hole.

Wire is a more convenient means of transmitting power than pipes or hose. Steam requires pipes with flexible joints for operating channeling machines. Steam pipes are often in the way and there is considerable loss of time in adjusting them in proper position. Compressed air may be transmitted through flexible hose, which is more easily adjusted than steam pipes, but transmission by wire is more convenient than either. If quarry operations are carried on in tunnels, electricity is especially desirable. A network of pipes is always a great hindrance to quarry operations.

Moreover, with electrical operation the percentage of loss in transmission can be determined and kept within small limits. With either steam or air there is a continual struggle with leaks, and radiation and transmission losses are usually excessive.

#### SOURCE OF ELECTRIC POWER.

If quarry operations are extensive and sufficient water power is within easy distance, it is probable that a hydroelectric plant owned and operated by the company is best, although the first cost may be high, especially if large dams must be built.

Uniformity of water supply is an important factor. If, during certain seasons, the water supply diminishes and it thus becomes necessary to construct auxiliary steam plants or to shut down operations for a part of the season, a hydroelectric plant is of doubtful advantage. Moderate variation in supply may be overcome by constructing a large reservoir.

In central Vermont three hydroelectric power plants supply most of the power required. Auxiliary steam plants supply additional power in times of low water. Transmission lines pass along the great marble valley for a distance of 62 miles. Power from various supply stations is turned into these lines and is drawn off at the quarries and mills scattered up and down its length. The power line is comparable with a great reservoir, into which water is pumped, and from which it is drawn off by pipe lines to supply various needs.

A Colorado marble company operates a hydroelectric plant with a 380-foot head of water. Pipes are laid up the mountain side and intercept streams at a high level. Consequently only small dams are required. Under such conditions a power plant can be established with a relatively small outlay of capital.

If electrical power can be obtained from a power company operating large central stations at a cost not exceeding  $1\frac{1}{2}$  cents per kilowatt-hour, it is probably better to purchase power than to erect a power plant at the quarry. If there are no available power lines, or if electrical power costs on the average more than  $1\frac{1}{2}$  cents per kilowatt-hour, and if the power required is 300 horsepower or



more, it will probably be more economical to erect a steam-driven generating plant at the quarry, provided the engine is of the condensing type. With coal at not more than \$3 per ton such a plant can be run economically.

At one quarry where the installation is of 500-kilowatt capacity with an average maximum load of 400 kilowatts, diminishing at night to 200, the total cost of power delivered on the main bus bars of the generating station is less than 1 cent per kilowatt-hour. Coal at this point costs a little over \$2 a ton.

The important point is that new quarries should adopt electrical operation from the outset if its adoption will be feasible at any stage of development. Current should be obtained in the most economical manner possible. Determination of the proper source of power often requires the advice of a consulting electrical and mechanical engineer.

#### COMPRESSED-AIR AND STEAM EQUIPMENT.

If for any reason electrical operation is inadvisable, operation by compressed air is unquestionably more economical than by steam, and vastly more convenient.

Direct steam power is undesirable on account of the transmission losses and the obstruction occasioned by the necessary network of pipes along the quarry floor. If steam must be adopted, care should be taken to reduce to a minimum the losses due to leakage and radiation.

As steam pipes are subject to varying temperatures and pressures, constant attention is necessary to avoid leakage. Plate III, *B*, shows a typical example of waste by leakage in a quarry transmission line. Many quarrymen fail to realize that money is pouring out of these leak holes at an incredibly rapid rate. A good pipe fitter should be on hand to remedy such conditions immediately.

Serious losses are also caused by radiation of heat from the steam pipes and by energy used to overcome the friction of the steam against the inner surface of the pipe. Such waste of energy may be minimized by properly covering the pipes with heat-insulating material. A few quarrymen have insulated their transmission pipes, but by far the greater majority have, as yet, made no attempt to do so. An additional means of reducing loss by condensation is the use of a superheater. Steam will not condense until it has lost its superheat, and consequently condensation of steam in pipes will not occur as long as any of the superheat remains. One degree of superheat is commonly conceded to carry steam 10 feet, but of course the net result depends on the diameter of the pipe and the condition of the insulation. If the factor mentioned be assumed, it may be con-

cluded that if steam is to be conveyed 1,000 feet 100 degrees should be added to its temperature in order that no condensation shall take place. In order to avoid as much as possible of this loss by condensation, the power plant should be situated close to the excavation.

If quarrymen could see the combined loss by leakage and radiation for a given period converted into a pile of coal they would realize that the loss assumes alarming proportions. One southern quarry company, using a battery of four 300-horsepower boilers, found by practical test that on a day of average temperature the full energy developed by one boiler was required to maintain the necessary steam pressure when no work was being done.

In old installations now operating with steam or compressed air any changes or additions ought to be determined with a view to the ultimate electrification of the entire plant. Neglect of this precaution may finally bring the operating company to a position where it can not compete with other companies that have electrified their plants.

### CHANNELING.

#### CHANNELING MACHINES.

Sullivan, Ingersoll-Rand, and Wardwell channeling machines are all used, and each type has its special advocates. For steam operation the Sullivan machine seems to be the favorite, whereas the Ingersoll electric air type is the favorite in quarries having direct electric connection.

The most suitable Sullivan machine for marble quarrying is probably the double-swivel channeler which can be used for straight vertical cuts, for undercutting, or for cutting out corners. A few quarries, in which operations are scattered over a wide area and in which electricity is not used, employ Sullivan machines with portable boilers attached. One Georgia company operates Sullivan "duplex" channelers, consisting of two machines on a single truck working in the same channel, as illustrated in Plate IV. For long cuts such channelers are undoubtedly advantageous, as one operator can manage two machines. For small quarries where the cuts are short and many corners must be cut one machine would probably be idle so great a share of the time that the advantage would be doubtful. These machines are commonly termed "double headers" to distinguish them from the Wardwell "duplex" channelers.

The electric air channeler is self-contained, all the mechanism being on the channeler truck. The air is compressed by a motor-driven "pulsator." The air is never exhausted into the open but is simply driven back and forth under pressure in a closed circuit. The machine may be used for vertical, inclined, or horizontal channeling. The roll guides between which the channel bits run are effective for

horizontal or inclined cuts. It is better balanced than most machines for cutting along the lower side of an inclined track.

It is claimed by some quarrymen that the electric air channeler will cut at least 10 per cent more than steam or air machines if all are equally well handled. This superiority is not due to the fact that the electric air channeler cuts faster while it is cutting, but there are fewer and shorter interruptions for adjustment, setting up, etc., so that a higher time efficiency is obtainable. Cables and wires are more quickly adjusted, are less in the way, and are more easily handled than steam pipes or hose. Also, many quarrymen think the electric air channeler is the most economical in power consumption of any machine yet devised.

The Wardwell "duplex" channeler cuts two parallel channels at the same time. On a level floor and with sound stock it gives good service. As pointed out later, in quarrying unsound stock it is sometimes desirable to vary the spacing of the channel cuts in order to make them coincide with joints. With the Wardwell machine such variation would not be possible, and in consequence it is not well suited for working in unsound marble. One company has successfully adapted this machine to electric operation.

Experiments in one company's quarries in North Carolina indicate that in channeling hard marble, light and rapid blows are more effective than slower and heavier ones. The change from heavy to light blows can be made in most machines by changing the stroke.

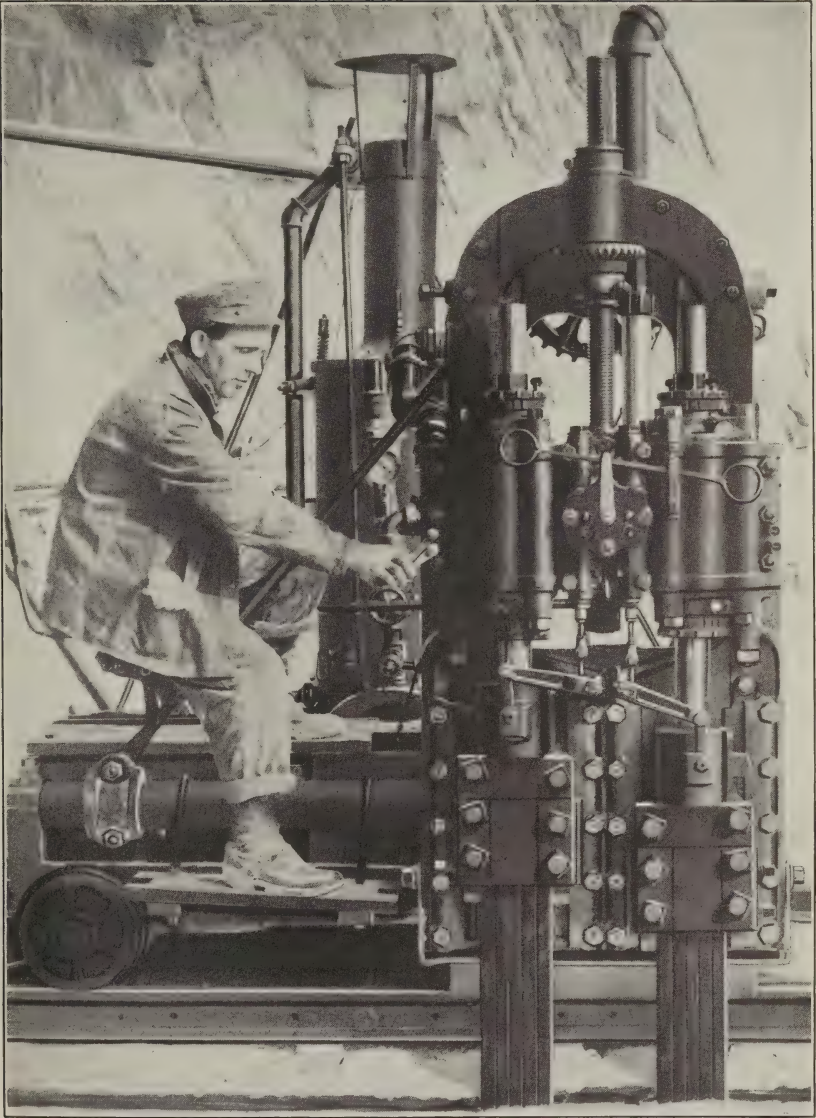
#### CHANNELING IN RELATION TO BEDDING.

##### INFLUENCE OF ATTITUDE OF MARBLE BEDS ON CHANNELING.

Most marble deposits occur in regions where great folding and contortion of the rock has taken place as a result of tremendous geologic forces. The original flat-lying beds may be tilted at all angles. This condition greatly complicates the process of quarrying. In numerous instances the beds are separated by open fissures which demand a process of cutting in conformity with them. In other quarries the beds may exhibit no planes of separation. A distribution of color or impurities parallel with the bedding may, however, constitute factors of equal importance in their influence upon the plan of rock removal.

If conditions are at all favorable, it is desirable to maintain a level quarry floor. Conditions may be such, however, that efficiency in quarry operation or reduction in the proportion of waste may demand that quarrying be conducted on an inclined floor. No absolute rules can be given, for, as pointed out later, the nature of the product and the uses for which it is to be employed have a direct influence on the most feasible quarry method.





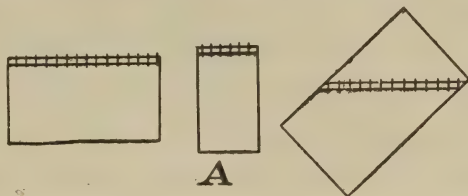
SULLIVAN "DUPLEX" CHANNELER.



LEVEL-FLOOR CHANNELING.

Figure 5, A, shows uniform rock with no open bedding planes and no decided rift. The beds may be horizontal, vertical, or inclined. This condition prevails in some of the Tennessee quarries. The bedding is so indistinct that it can be recognized only by the characteristic "crow feet," as otherwise the rock is uniform throughout. Under such favorable conditions, channeling may be carried on with a level floor, whatever the attitude of the beds may be.

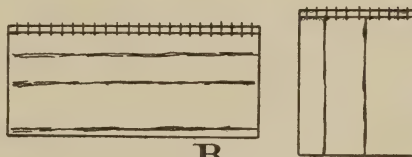
Figure 5, B, shows marble with open bedding planes, the beds being either horizontal or vertical. In either case a level quarry floor may be maintained.



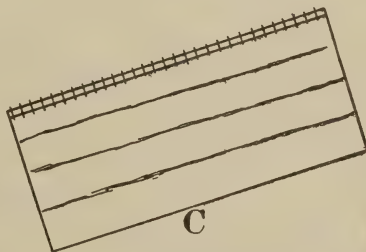
A

SAW-TOOTH FLOOR CHANNELING.

Figure 5, C, illustrates open marble beds, inclined at an angle less than  $45^\circ$ , such as are worked in a number of Tennessee, Alabama, and Vermont quarries. It is customary in some of the quarries to excavate right-angled blocks and to remove a row of blocks from each successive bed, resulting in the formation of a saw-tooth quarry floor as illustrated in figure 6.



B



C

FIGURE 5.—Proper positions of channeling machine tracks for beds of different inclinations. A, horizontal, vertical, or inclined beds that have no open bedding planes or decided rift; B, horizontal or vertical beds separated by open bedding planes; C, open beds inclined at an angle less than  $45^\circ$ .

In quarrying a saw-tooth floor the channeling-machine track is supported by timbers as shown in Plate III, C. The disadvantages of such a method are well known to many quarrymen, but are accepted as inevitable because no better method has appealed to them. The disadvantages may be enumerated, as follows: (1) The construction of supports to hold a track in a horizontal position, perhaps 6 or 8 feet above a slanting rock surface, requires considerable timber and many hours of labor, part of which is done by skilled men. (2) There is a great loss of time in starting and straightening channel cuts. The machine is so high above most of the rock surface that the channel bits, no matter how tightly they are clamped,



swing to one side or spring when they strike the rock, so that starting or maintaining a straight cut is difficult. In practice the helper generally takes a hand tool and hammer and starts the cut. After a little progress with the machine further delay may be necessary to straighten the cut. Altogether much time may be lost before the machine may be worked continuously. (3) If the rock has a steep dip a short length of track only may be set up, usually resulting in loss of efficiency. In general, operations on a small scale are less economical than those on a large scale. A short channel cut is more expensive per square foot than a long one under similar conditions. (4) On account of the rapidly increasing height of the track above the rock surface as the machine travels in the direction of dip, one length of steel can be used over a small part of the course only, and hence short cuts with frequent changes of steel are necessary.

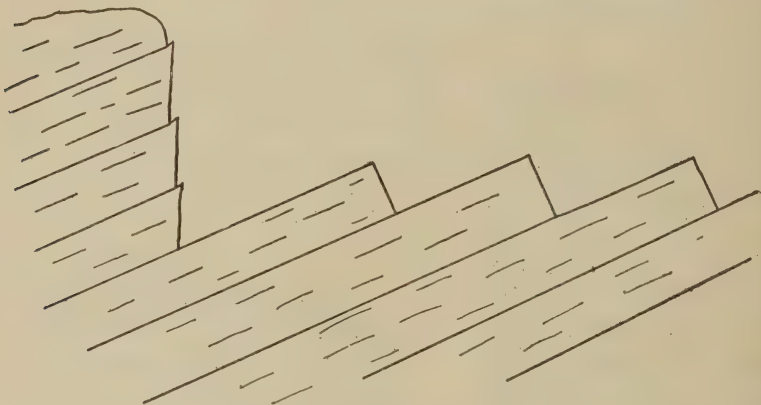


FIGURE 6.—A "saw-tooth" quarry floor.

(5) The necessity of cutting out numerous corners may increase the proportion of lost time.

#### INCLINED-TRACK CHANNELING.

An improved method of quarrying on an inclined floor is to place the channeling-machine track on the inclined rock surface in the direction of the dip. If the floor is of the saw-tooth type, the upper beds should be worked down successively until a large area of rock surface is obtained parallel with the open beds, as illustrated in figure 7. The force of gravity, which would cause the channeling machine to run too rapidly down the grade and probably entirely prevent its ascent is overcome by the use of a balance car or balance weight. The balance car may be placed on a parallel inclined track and loaded with enough stone or iron bars to exactly counterbalance the weight of the channeling machine, to which it is attached by means of a cable running over a sheave. The sheave is attached to

the upper end of the track. This method is employed with success in several Vermont quarries.

A second method is to place the balance-car track on a slanting unused part of the quarry. Though requiring a long cable attachment the plan works well, and is used successfully in Alabama. There is a considerable saving of time and expense in having a permanent balance-car track.

A third device, which must be arranged with special care in order to avoid the danger of accident, is a counterbalance weight attached to a cable that passes over a sheave in the tunnel ceiling.

For light grades a hill-climbing device observed on some machines gives satisfaction. It consists of a cable that passes over a sheave at the upper extremity of the track and winds on drums on the truck axles. The device prevents the machine from slipping, but does not equalize the load like the balance car.

#### HIGH EFFICIENCY OF INCLINED- TRACK CHANNELING.

Quarrymen who have not had the experience may doubt the success of inclined channeling. However, many who have tried it have met with unqualified success. One Alabama

company claims that channeling on a floor parallel with bedding that dips about  $33^{\circ}$  has resulted in a 50 per cent increase in efficiency over the saw-tooth method of operation.

In this connection the following quarry report of a company operating in Quebec, Canada, is of interest:

#### *Monthly channeling report covering work of day gang in a Quebec quarry.*

Machine No.	Marble excavated.	Time in operation.	Time moving.	Time under repair.	Time shut down.
	<i>Square feet.</i>	<i>Hours.</i>	<i>Hours.</i>	<i>Hours.</i>	<i>Hours.</i>
3.....	529.6	139.5	44.5	34	31
5.....	613.0	157	31	31	32
7.....	695.5	208.5	26.5	-----	29
8.....	712.5	187	23.5	18.5	24
9.....	1,355.1	178	44	8	23

The good showing of machine 9 is the notable feature of this report. Machines 3, 5, 7, and 8 were operated on level elevated tracks on a saw-tooth floor. Machine 9 ran on an inclined track with a balance car. The average month's cutting per machine by the saw-tooth method was 637.6 square feet, whereas the one machine with the

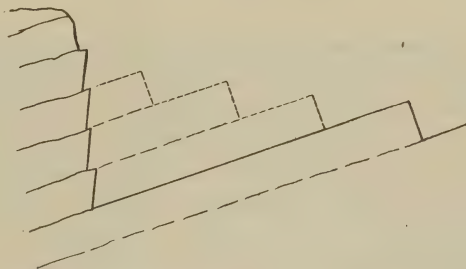


FIGURE 7.—Manner in which an inclined floor may be developed from a "saw-tooth" floor.

balance car cut more than double that amount. Such figures offer convincing proof of the efficiency of inclined channeling.

With many quarrymen the physical difficulties of moving and operating machinery on inclined floors constitute sufficient cause for rejecting the method. However, it has been found that when men become accustomed to the new methods they work with the same facility as on level floors, and may even prefer the changed conditions. Methods of employing cables, snatch blocks, and hoists for handling heavy machinery are quickly devised by intelligent foremen. In one Vermont quarry inclined channeling is conducted successfully on a floor slanting  $45^{\circ}$ . The cost of channeling is approximately 14 per cent greater than on a level floor, and the cost of other operations shows a small increase.

When channeling machines are being operated on inclined tracks, no scaffolding is required. The tracks are placed flat on the rock surface with little more expense than when placed on a level floor. Much time is saved in the process of starting and straightening cuts, the field of operations need not be limited by short tracks, and the work is not interrupted by frequent changes of steel.

Plate V, A, shows a marble quarry with open beds dipping  $24^{\circ}$  to  $28^{\circ}$ . The present method is to channel in short cuts with the channeling machines working on tracks supported in level position. The inclined floor shown in the plate has an area of about 4,000 square feet which is sufficient to warrant the use of the inclined track and the balance car.

Even if conditions seem unfavorable the inclined-track method may prove to be the more economical. In one quarry observed the structure is a low anticline or arch, and the open bedding planes dip westward in the western part of the quarry, maintain an approximately horizontal position near the center, and dip eastward in the eastern part. The rock is now quarried on a level floor with the production of a large number of angular blocks. It is probable that the maintenance of a quarry floor parallel with the bedding, even though it involved varying angles of inclination in different parts of the quarry, would give better results.

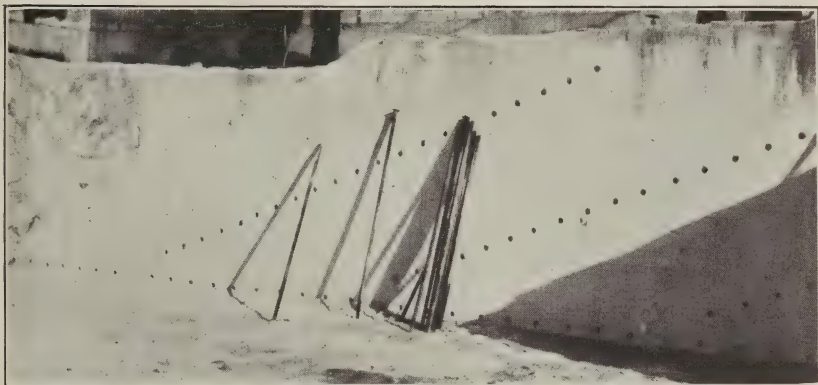
#### INCLINED-TRACK CHANNELING ACROSS BEDS.

\* Figure 8, A, illustrates open marble beds dipping at an angle greater than  $45^{\circ}$ . In order that right-angled blocks may be obtained without a saw-tooth floor, the obvious method of channeling is to make the floor perpendicular to the beds as indicated in the figure. Such a method has not been observed in any American quarry, though it should be successful.





A. A QUARRY FLOOR SUITABLE FOR INCLINED-TRACK CHANNELING.



B. METHOD OF QUARRYING ACUTE-ANGLED BLOCKS.



C. CHANNELING DIAGONALLY ON A QUARRY FLOOR INCLINED  $36^{\circ}$ . A BALANCE CAR IS USED TO COUNTERACT THE EFFECT OF GRAVITY. QUARRY AT GANTTS QUARRY, ALA.



## INFLUENCE OF RIFT OR COLOR BANDS ON CHANNELING.

Figure 8, B, illustrates marble beds dipping at an angle of less than  $45^\circ$  but with no open bedding planes. They are assumed to have, however, parallel with the bedding, a series of color bands or streaks of impurities or a decided rift, or the successive beds are of varying quality. The most desirable development of such beds is more complicated.

If, for artistic effect, it may seem desirable to have the color bands pass diagonally through the blocks, it may be advisable to excavate right-angled blocks on a level floor. Ordinarily, however, a proper classification of material would demand that the marble be split parallel with the bed, also if the material is uniform in quality, and possesses a decided rift, it can be worked more easily parallel with the bedding.

Figure 8, C, illustrates beds having rift, impurities, or color bands as represented in the beds shown in figure 8, B, but with the beds

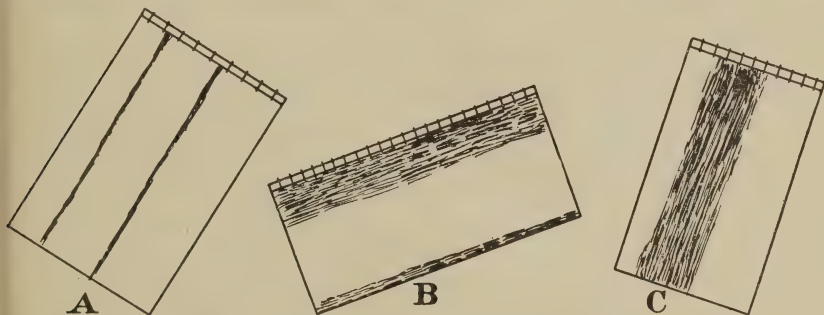


FIGURE 8.—Proper positions of channeling machine tracks for beds of different inclinations. A, open beds inclining at an angle greater than  $45^\circ$ ; B, nonuniform beds inclining at an angle less than  $45^\circ$ ; C, nonuniform beds inclining at an angle greater than  $45^\circ$ .

dipping at an angle greater than  $45^\circ$ . In developing such beds, if it seems advisable to use inclined channeling, the floor should be at right angles to the beds rather than parallel with them.

## WASTE DUE TO ACUTE-ANGLED BLOCKS.

If steeply inclined marble beds are channeled on a level floor, and the cross breaks or cross channel cuts are made parallel with the bedding, acute-angled blocks will result, as is illustrated in Plate V, B. A complete discussion of the conditions under which such blocks may or may not be quarried economically is given in a later section dealing with the problem of waste. As pointed out in that section, if thin stock is desired, the waste resulting from acute-angled blocks may not be excessive. Saw gangs require longer blades however, and when longer blades are used the tendency toward unevenness in



the cuts is increased. If much cubic stock is desired undoubtedly it is more economical to quarry on a slanting floor parallel with the beds.

#### EXAMPLE OF INCLINED CHANNELING.

A large tunneled quarry operated in Vermont furnishes a good illustration of quarrying in conformity with bedding, and a detailed description of the various processes may be instructive.

Near the top of the quarry the beds dip about  $70^{\circ}$  east. At a depth of 50 or 60 feet, they curve back and dip west, and lower down again dip to the east, the angle of dip decreasing to about  $30^{\circ}$  near the bottom of the quarry. There are few open bedding planes. The excavation is about 300 feet deep, and long tunnels are driven in the direction of the strike.

Gray or variously colored bands occur at intervals, marking off distinct beds. The beds vary greatly in quality, some being almost pure white, and others gray, wavy, or banded. The various beds are named, and the characteristics and qualities are well known to the foremen.

In the upper part of the quarry, where the dip of the beds is steep, a level quarry floor is maintained. According to the rules given above, the quarry floor in this part of the quarry should be inclined to cut across the beds at right angles and thus avoid the production of acute-angled blocks. In this particular place, however, a level floor is preferable. As mentioned above, the beds near the top of the quarry dip to the east, at a lower level are vertical, and at still greater depth dip west. If the floor were to be maintained at right angles to the bedding the condition described would necessitate a constantly changing floor level, resulting in the production of many angular blocks.

Except for wall cuts, channels are made in one direction only—at right angles to the strike. The long masses of rock thus obtained are divided into small blocks by wedging in drill holes, projected in rows parallel with the bedding, as shown in Plate V, *B*. Effort is made to so place the holes that each block is of uniform quality.

Near the bottom of the quarry, where the dip of the beds is  $45^{\circ}$  or less, the quarry floor is maintained parallel with the bedding. A balance car on a parallel track is employed to overcome the effect of gravity, and thus permit the channeling machine to operate with the same motive power as on a level track.

The depth of the channel is governed by the position of those natural lines of separation that mark out a definite change in the quality of the rock. The channel cut may pass through one or two of these lines of division, but is always terminated by one of them.

If no open bedding planes appear, each of the bands of uniform material may, for convenience, be designated as a bed. The method of determining when the bottom of a bed is reached is of interest. The channeling machine, when running in its ordinary vertical manner, continually loses space at the end of its run, leaving a slanting end. Water is poured down this inclined end in order to wash it clean, and an incandescent light with a specially flattened bulb is let down. By this means, one who is thoroughly familiar with the rock may identify at a depth of 10 to 12 feet the bands that separate the various beds.

When drilling or channeling is done at right angles to the quarry floor, right-angled blocks are obtained. By working thus on an inclined floor, blocks of economical form are produced, and an exact classification of the product is made possible.

#### INFLUENCE OF DIP AND STRIKE ON DIRECTION OF CHANNEL CUTS.

The influence of bedding on the attitude of the quarry floor has been discussed in detail. The influence of dip and strike on the direction of the channel cuts is also of great importance. If vertical channel cuts make oblique angles with the strike of a marble deposit in which separation of blocks must be made along the planes of steeply dipping beds, it is obvious that blocks having too much of an acute angle will result. It is generally more convenient and more economical to make channel cuts parallel with the strike. Exception may be made if pronounced joint systems meet the strike of the rock at oblique angles. Under such circumstances channeling parallel with the joints rather than with the strike is justifiable. However, an inclined floor should be maintained, and the cuts should be at right angles to the floor in order that rectangular blocks may be produced.

#### CHANNELING IN RELATION TO UNSOUNDNESS.

##### CHANNELING PARALLEL WITH JOINT SYSTEMS.

In a previous section (pp. 22-29) a discussion is given of the causes of unsoundness and of the various forms in which it appears. The most important feature of joints in relation to channeling is their occurrence in more or less definite systems. The importance of recognizing such systems and quarrying in accordance with them can scarcely be overestimated. In quarries in which joints are prominent the quarrymen should endeavor to make their channel cuts parallel with the chief joint systems. Blocks that are intersected by oblique joints are almost useless. If, on the other hand, the joints parallel one pair of faces, the waste is greatly reduced.

## CHANNELING COINCIDENT WITH JOINTS.

Paralleling the joint system is only one step in economical channeling. If it be assumed that the joints intersecting the blocks are parallel with one pair of faces, considerable waste may still result. If saw cuts are made parallel with the joints, two or three slabs only may be wasted. Usually, however, joints cut across the grain and slabs are cut parallel with the grain. A single joint, therefore, will intersect every slab. Consequently, economy demands that channel cuts not only run parallel to joints, but that they be spaced in such a manner

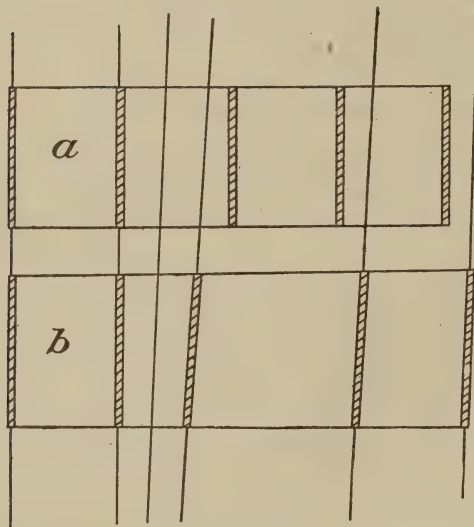


FIGURE 9.—The right and the wrong way of channeling unsound marble: *a*, Channel cuts equally spaced, involving waste; *b*, channel cuts made coincident with joints, avoiding waste.

as to reduce to a minimum the number of joints contained within the blocks. In other words, whenever possible the channel cuts should coincide with joints. If joints are spaced at irregular intervals, channel cuts should be spaced in the same way. It is unreasonable to maintain that the production of blocks varying in size is undesirable. The presence of irregularly spaced joints will not permit the quarrying of uniform sound blocks, and it is clear that in the shaping of any structural design sound blocks of various

sizes will cut to better advantage than unsound blocks of uniform size.

The advantage of eliminating joints by channeling coincident with them is illustrated in figure 9. Six unequally spaced joints are shown in an approximately parallel system. In the figure *a* represents equal spacing of channel cuts without regard to unsoundness. Of the four blocks that result, one is intersected by one joint and another by two joints, and two blocks are sound. At *b* is shown spacing varied in such a manner as to make the channel cuts fall on the joints. As a consequence one small block is intersected by a joint, and three blocks, one of which is exceptionally large, are sound. A judicious spacing of channel cuts may thus tend to produce a large proportion of sound stock.



## ECONOMY OF CHANNELING IN ACCORDANCE WITH UNSOUNDNESS.

Figure 10 shows the economy of channeling in accordance with unsoundness. Two right-angled systems of unequally spaced joints are shown. They are identical in both parts of the figure. As shown

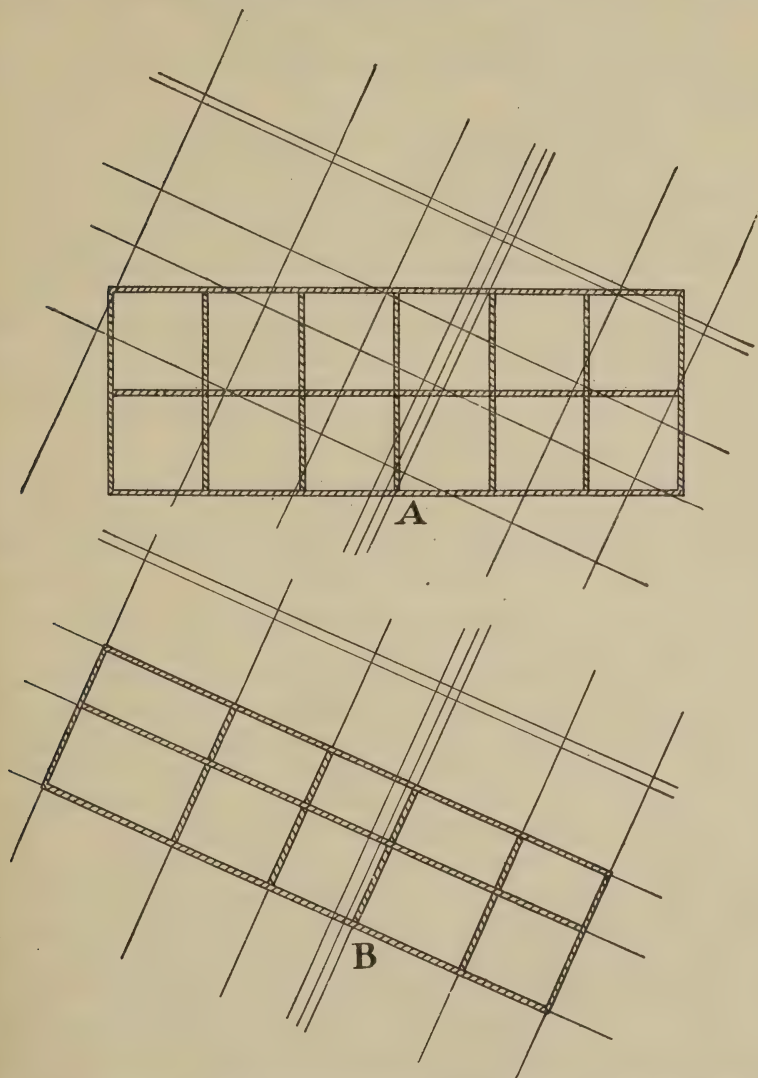


FIGURE 10.—The economy of channeling in accordance with joint systems: A, Plan of channeling without reference to direction or spacing of the joints, involving great waste of marble blocks; B, plan of channeling in which channel cuts are made parallel with, and as far as possible coincident with, joints, avoiding waste of blocks.

in figure 10, A, the channel cuts make oblique angles with the joint systems. The channel cuts are spaced regularly in both directions without regard to unsoundness. As a consequence, every block quarried is intersected by one or more joints.

Figure 10, B, illustrates the remarkable improvement that results from changing the direction of channel cuts to parallel the joints, and from varying the spacing of the cuts so that they fall exactly on the joints. Of the 10 blocks that will be produced 8 will be sound and 2 will have two parallel joints close to one side. The advantage of cutting out blocks in accordance with unsoundness is so apparent that it is remarkable how few quarrymen have made practical application of the principle.

#### UTILIZATION OF JOINTS FOR MAKING CROSS BREAKS.

Channeling may be done in one direction only, the cross breaks being made by some other means, or the rock may be channeled in two directions. In many marble deposits, one system of joints is pronounced, and cross joints are few in number. Under such conditions, it is wise to channel in one direction only, at right angles to the chief joint system. Advantage may thus be taken of the joints in making cross breaks. A long mass of marble can be channeled at the back and ends, and wedged up at the floor until it is free. The wedges near one end may then be driven hard and the others left untouched, producing a bending strain which may be sufficient to cause the mass to break at one of the joints, which is a plane of weakness. Heavy wedging at both ends, the middle wedges being left untouched, may be even more effective.

#### AN ILLUSTRATION OF CHANNELING IN ACCORDANCE WITH UNSOUNDNESS.

An excellent illustration of a modification of channeling directions to parallel strike, dip, and unsoundness is to be found in a certain quarry in Vermont. One channel cut parallels the strike, which is N.  $5^{\circ}$  W. The beds dip about  $70^{\circ}$  to the east, and, on account of a green banding parallel with the bedding, the proper classification of the material and the production of attractive patterns in the banding demands that the channel cuts be inclined to parallel the beds. A remarkable system of parallel open joints (see fig. 2, p. 26) intersects a limited part of the quarry. The joints run N.  $82^{\circ}$  E. and dip  $61^{\circ}$  N. The second channel cut is made parallel with these joints. Thus the channel cuts meet at an angle of  $87^{\circ}$ , and are both inclined from the vertical. The maintenance of a level quarry floor is justified by the fact that such conditions prevail over part of the quarry only. By thus quarrying in accordance with rock structures, sound blocks with a desirable color distribution are obtained.

A short distance north of this quarry is another, which is now about 50 feet deep. The system of joints referred to above does not appear as yet in the excavation. As the joints dip  $61^{\circ}$  N., it is an easy matter to determine the point at which, with continued excavation, they

may be expected to appear. The present plan of quarrying is to channel vertically at right angles to the strike, and to make the cross breaks by wedging in drill holes parallel with the bedding. When the zone of parallel joints is reached, a modification of the method will be justified.

#### A PROPOSED IMPROVEMENT IN QUARRY METHODS.

As another illustration of quarrying in conformity with unsoundness, reference may be made to a certain marble quarry in which a change of plan is contemplated. The prevailing joints run north and south, and a second series of joints filled with crystalline calcite, known locally as "glass seams," run east and west.

The walls of the present excavation run approximately northeast and northwest. Thus a majority of the joints intersect blocks diagonally. When the present excavation is extended it is proposed to make the cuts in north-south and east-west directions, and thus parallel the chief rock structures.

#### INDISTINCT JOINTS AND JOINT SYSTEMS.

In the preceding paragraphs reference has been made only to those joints or joint systems that are easily recognizable. In quarries in which joint systems are obscure, the problem of economic channeling is more difficult.

The difficulty of recognizing joint systems is due to two main causes. The joints may be rather easily seen, but so irregular and intersecting at so many angles that system seems to be absent, or they may be so indistinct that only skilled men with long practice can recognize them in the quarry. When joints are both indistinct and irregular the problem is complex.

#### HOW TO DEAL WITH COMPLEX JOINTING.

A desirable method of approaching such a complex problem can be best appreciated by consideration of a concrete example. The operations of an Alabama marble company offer one of the best illustrations obtainable of the means that may be employed to discover systems of unsoundness and of how quarry methods may be modified later in accordance with these systems.

The quarry operations are greatly hampered by the presence of cracks that are locally termed "headers," many of which are almost unrecognizable in the quarry. However, when the blocks are sawn they cause the resulting thin slabs to break into small and angular pieces, many of which can not be used. So serious was this difficulty that profitable operation seemed impossible.



The marble was first quarried on a saw-tooth floor parallel with and at right angles to the strike. The beds dip about  $33^{\circ}$ . Blocks were intersected by joints and also by mica bands which run parallel to the bedding, and the waste was excessive. The proportion of sound stock was increased to some extent by quarrying on an inclined floor parallel with the bedding. Even with this improvement the presence of many "headers" intersecting the blocks obliquely resulted in great waste.

In an attempt to further overcome the difficulty the quarry floor was laid out in sections, and the marble blocks as they were quarried were all so numbered and oriented that the section to which they belonged and their exact position in the section could be determined. The visible "headers" were located accurately, and diagrams were made showing their position. The "headers" that could not be located in the quarry were found in the slabs after the blocks had been sawed. As every block had been oriented and as the original position of the slab was known, reference of the slab to its original position in the quarry indicated the point at which the joint must cut the quarry wall. As it was known that the joints must be present at certain points, diligent search was made to find evidences of them on the quarry walls. Only persons skilled in such observation through long practice can recognize obscure joints. In the quarry under discussion one efficient quarryman is detailed for the express purpose of finding and marking them on the quarry walls and floors.

The information thus obtained was supplemented by study of a drill core obtained by projecting a hole down the dip with a double core-barrel drill. A diagram of all the "headers" thus discovered on one of the most unsound floors is shown in figure 11. As may be seen from this figure, the joints in general follow two distinct systems. These two sets of joints have for convenience been designated  $x$  cracks and  $y$  cracks. The general direction of the  $x$  cracks is  $S. 80^{\circ} E$ . They are distinctly visible, regular, and sufficiently far apart to allow the production of fair-sized blocks. They are all normal faults, having a slip of less than 1 inch. The  $y$  cracks are those mentioned above as being very indistinct. Their direction is  $N. 24^{\circ} E$ . They are closely spaced, branching, and fade out and reappear irregularly. They are not faulted, or at least there is no apparent slip. They occur in zones about 50 feet wide, and these are followed by zones 60 feet or more in width in which the  $y$  cracks are absent or few in number.

In addition to the  $x$  and  $y$  cracks there is a series termed "slick seams," which run in the direction of the dip. They are probably caused by expansion and contraction from solar heat. They disappear at a depth of about 75 feet. A few joints are noted in the direc-

tion N.  $82^{\circ}$  E. The strike of the rock is N.  $46^{\circ}$  E., and the dip averages about  $33^{\circ}$  SE.

As indicated in figure 11, the two main systems—the  $x$  and the  $y$  cracks—are  $76^{\circ}$  apart, and the  $x$  cracks make an angle of about  $54^{\circ}$  with the strike.

With this arrangement of headers, four plans of channeling were possible, as follows:

- (1) Channels parallel with and at right angles to the strike.

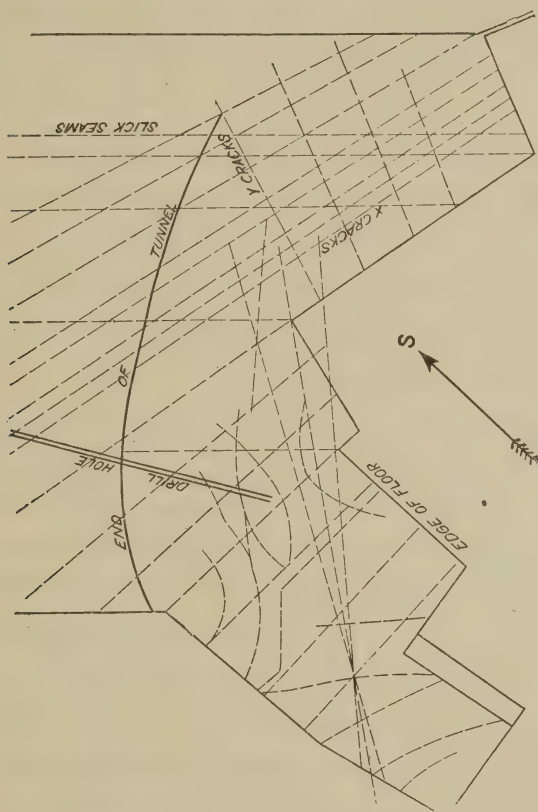


FIGURE 11.—Diagram of joint systems in one bed of an Alabama marble quarry.

- (2) Channels parallel with and at right angles to the  $y$  cracks.
- (3) Channels parallel with and at right angles to the  $x$  cracks.
- (4) Channels parallel to the  $x$  and  $y$  cracks.

Worked out mathematically, plan 4 is the best, but only slightly better than plan 3. Where quarrying is now carried on, the  $y$  cracks are not abundant, and plan 3 is now followed. If the  $y$  cracks become more numerous with further development, plan 4 may be adopted.

The conditions prevailing on this floor were exceptionally bad. With the most economical methods only 15 per cent of the marble quarried was in sound blocks. On the next floor below 30 per cent was sound, and on the third floor 35 per cent. Quarrying parallel with and at right angles to the strike on the floor mapped would give no sound blocks, and probably few would result from a similar method on the lower floors.

It is established beyond question, therefore, that the most economical method of quarrying in this deposit is to channel diagonally on a slanting floor. To quarrymen who would hesitate at quarrying on an inclined floor under favorable conditions such a method must seem impractical. However, its success in this particular quarry is assured. Although the difficulty of operation is undoubtedly great, the advantages more than offset the increased cost per square foot of channeling. Plate V, *C*, illustrates a channeling machine operating on a track placed diagonally on an inclined floor.

The conditions in the quarry mentioned above are about the most complicated that can occur, as the joints are not only seemingly irregular, but many of them are indistinct. If the most complicated conditions can thus be handled satisfactorily simpler problems should be comparatively easy of solution.

#### CUTTING WITH UNSOUNDNESS IN TENNESSEE QUARRIES.

In some of the Tennessee quarries there are distinct joints known as "cutters," and other cracks which are indistinct, appearing as fine white lines. The latter are known as hair lines. Though the hair lines are difficult to recognize in blocks or in the quarry, thin slabs break readily along these planes.

In some quarries the distinct cutters show a prevailing system and the hair lines a second system nearly at right angles to the former. Under such conditions the rock may be quarried to advantage if the cracks are not too close together.

An excellent example of quarrying in accordance with rock structure and unsoundness is to be found in a quarry near Knoxville, Tenn. Open bedding planes dip 15° to 20° N., "cutters" occur in a predominating east-and-west system, and indistinct hair lines run generally north and south. The rock is channeled north and south, drilled east and west, and quarried on a slanting floor, thus paralleling the three prominent rock structures.

#### POSSIBLE OBJECTIONS DISCUSSED.

Some quarrymen would condemn a new method immediately if it were found that a decrease in the rate of channeling resulted. Such action would indicate faulty reasoning. Quarrymen must realize



fully that their purpose is not to produce channel cuts and drill holes, but blocks of sound marble. If by a new method a greatly increased proportion of sound stock is obtained at a nominal increase in cost of production, the method is successful even though the rate of channeling is greatly reduced.

To properly search out and map unsoundness requires considerable time and energy. However, the far-seeing quarryman realizes that such labor, although temporarily unproductive, prepares the way for more economical and efficient operations. It is important that quarrymen should recognize the undoubted desirability of such preliminary investigation and the modification of methods to suit the conditions.

#### METHODS OF SEARCHING FOR INDISTINCT JOINTS.

If joints are very indistinct, a method of search devised by a Tennessee marble company is noteworthy. As all quarrymen know, only distinct joints can be recognized on a channeled surface. They are much more easily recognized on a fractured surface. Hence, before making cross breaks on a long channeled mass the quarryman takes a hand tool and a hammer and "points" a strip about 6 inches wide in a horizontal direction across the channeled surface. The effect is to produce a band having a fractured surface on which the natural joints are easily detected. The cross breaks are then made where the joints appear, and much of the unsoundness is in consequence eliminated from the finished blocks.

The same method is followed on the quarry floor. "Pointed" bands are made in two directions at right angles and 10 to 12 feet apart. The joints are thus located and the channel cuts can then be made to best advantage.

In a certain Alabama quarry small pieces are broken off the corner along the working edge of a quarry floor, thus exposing fractured surfaces. When water is thrown on the surfaces the indistinct joints appear as fine white lines. Channel cuts are then spaced to coincide with the joint lines wherever possible.

#### THE BOWLDER QUARRIES OF TENNESSEE.

Unsoundness in certain Tennessee marble quarries is greatly emphasized by erosion. Running water has entered the minute fractures and so worn them that large cavities have been formed between which the rock stands up as pinnacles and spires. The cavities are now filled with clay. Such a condition constituted what is known in Tennessee as a boulder quarry. As the erosion began in planes of weakness it is probable that the masses of rock that now remain are comparatively sound. In such quarries little if any channeling is necessary.

•



wall cuts,  $x$ ,  $y$ , and  $z$ . Next, the two cuts,  $m$  and  $n$ , are made. Their purpose is to remove the necessity of cutting out numerous corners. By making these cuts only three corners,  $a_1$ ,  $a_2$ , and  $a_3$ , must be cut out. After the cut  $o$  has been made, the short cuts  $b_1$  to  $b_8$  follow, and finally the cuts  $c_1$  to  $c_8$ . The key blocks 1 to 10 are broken free successively by gadding and wedging, beginning at the open edge of the bench. When these blocks have been removed, blocks 11 to 19 are successively raised and removed. It will be noted that the prevailing joints run approximately east and west. Blocks 11 to 19, when wedged up from beneath, break along joints. This plan has the double advantage of making cross drill holes unnecessary and at the same time of reducing to a minimum the number of joints that intersect blocks. Blocks of considerable length free of joints may be used for columns.

#### CHANNELING IN RELATION TO RIFT.

In certain marble deposits there is a tendency for the rock to split with greater ease in one particular vertical plane. Ordinarily, under such conditions, channel cuts should be made at right angles to the direction of the rift in order that advantage may be taken of this ease of splitting, thus facilitating the making of cross breaks. If no rift exists, the difficulty of obtaining a straight and even break may be so great as to justify channeling in both directions.

#### USE OF WIRE SAW IN QUARRIES.

##### MECHANISM OF THE WIRE SAW.

The wire saw consists of an endless wire rope about one-fourth inch in diameter and composed of three strands. The wire passes around a driving wheel and is carried on pulleys to that part of the quarry where cutting is to be done. A sliding carriage is placed at a convenient point in the circuit to give the necessary tension to the wire. Renwick<sup>a</sup> states that in France and Belgium holes 2 to 3 feet in diameter are cut by means of cylindrical core drills. The cores are removed and standards erected in the holes. Pulleys over which the wire runs are attached to these standards.

##### THE PENETRATING PULLEY.

In early days much difficulty was experienced in guiding the wire in a slanting direction, but this difficulty was overcome in 1898 by the invention of the "penetrating pulley." This device is thus described by Renwick:<sup>b</sup>

The pulley consists of three parts—(1) the pulley: This is a steel disk 19.6 inches in diameter and 0.27 to 0.31 inch thick, grooved on its edge to receive the wire. On

<sup>a</sup> Renwick, W. G., *Marble and marble working*, London, 1909, p. 34.

<sup>b</sup> Renwick, W. G., *Op. cit.*, p. 38.



the central part of the disk and projecting on each side is a boss supporting a steel axle 0.97 inch long. (2) The fork: This takes the shape of a hollow steel bar 2.4 inches in diameter, grooved at the lower end to receive the pulley, and which can be lengthened by a series of tubes of similar diameter. The bottom of the fork acts as a bearing, on which the pulley runs. (3) The carriage: This consists of a standard which can be placed in position by three screws or guys. The standard carries a screw and drum attachment, by which an automatic progressive motion is given to the fork from the rotation of an exterior pulley, on which the helicoidal wire runs.

The operation of extracting stone is as follows: Two holes are first sunk to receive the fork on which the pulley runs. The standards carrying the wire having been placed in position and the wire set in motion, the penetrating pulley is brought close to the rock. The thickness of the pulley being slightly less than the diameter of the wire, the latter projecting from the edge of the pulley wears into and bites the rock, forming a groove into which the pulley enters. Sand and water are fed to the descending wire, and, the fork supporting the pulley following the hole made to receive it, the wire is carried through the rock from one standard to the other, making a cut in the direction required. Crosscuts are obtained by altering the position of the standards. Cuts can be made along a length of 10.9 to 16.3 yards, and for an equal depth, the progress of the cut being 1.56 to 5 inches per hour, this depending on the nature of the material and the length of the cut.

#### INSTANCE OF USE OF WIRE SAW IN QUARRY.

Although the wire saw is used extensively by marble-finishing plants, its use in American quarries has, during recent years, at least, been confined to certain Colorado quarries. In these quarries it has been found that the most useful application of the wire saw in quarry work is in the cutting out of masses of rock situated between two shafts. In the early days of these quarries three shafts 60 to 80 feet apart were opened in the mountain side. After they had been carried down to a considerable depth it was found advantageous to remove the intervening masses and to develop a single large opening. By means of a core drill a hole was projected from one excavation to the next at a point close to the ceiling. The wire was passed through this hole, fitted around the necessary pulleys, and spliced to form a continuous belt. Steel shot and water were used as abrasive.

A mass 4 feet wide and 5 feet high and extending from one opening to the other was first removed to give a working space. Both wall and ceiling cuts were made with the wire saw. The ceiling cuts, were somewhat slower and more difficult to make than the wall cuts, as the abrasive would not feed readily to the saw. A second horizontal cut was made about 18 inches below the ceiling cut. The intervening mass was broken up and removed as waste in order to gain sufficient room for the removal of blocks without jamming. On account of the narrowness of the saw cuts, great difficulty is often encountered in removing this material. It may be noted, however, that a great saving of marble is thus effected, as in ordinary tunneling the preliminary opening is 6 or 7 feet high. Additional vertical cuts were

spaced in the same manner as those made with a channeling machine, whereas the cross and the floor breaks were made by drilling and wedging.

It has been found that the cutting is just as effective if the abra-sive is fed to the saw at one point only, the point where it enters the rock, as when fed at several points along its course through the rock.

#### DESIRABILITY OF USING LONG WIRE.

It is important that a long wire be employed. The wire is abraded during the process of sawing and continually becomes smaller until at some point it becomes too weak to withstand the strain and breaks in consequence. It is obvious that by increasing the length of the wire the rate of wear on any given part is correspondingly diminished. One company uses a wire nearly a mile long. Formerly the super-fluous wire was passed around a drum in numerous turns. It has been found more satisfactory, however, to run the wire in a simple turn over a pulley situated at a distance from the quarry.

#### METHOD OF REPLACING AN OLD WIRE BY A NEW ONE.

In the early days of wire-saw operation much time was lost when a wire was worn out before a cut had been completed, as the opening cut by a wire gradually becomes narrower as the wire is reduced in size by abrasion. When the wire was replaced by a new one much of the cutting had to be repeated, as the opening was too narrow for the new wire.

Recently a method has been employed that obviates this difficulty. When the wire has been worn small and appears nearly ready to be discarded, the force that crowds it against the bottom of the cut is relaxed, and it is allowed to run freely for some time. It thus wears an opening larger than its diameter. The new wire can be drawn through this opening, and the cut continued with little loss of time.

#### DISADVANTAGES IN USE OF WIRE SAW.

When working normally the wire saw cuts much more rapidly than a channeling machine and requires less attention.

The presence of flint balls in marble is, however, a serious obstacle to the successful operation of the wire saw. On account of its extreme hardness the flint will greatly diminish the speed of cutting or may entirely suspend progress. Moreover, the wire is likely to pass around the mass of flint, and in doing so, may be offset 3 or 4 inches from its regular course, resulting in an uneven surface.

Another disadvantage in the use of the wire saw is the obstruction of other operations by the wires that pass through the quarry.

They interfere more particularly with the pulling and the hoisting of blocks. Seemingly, the wire saw is best adapted for cutting out masses of rock that intervene between two shafts. Its advantage over the channeling machine in the ordinary process of quarrying has not yet been demonstrated in any American quarry.

### DRILLING.

### MACHINERY.

Ordinarily in quarry drilling vertical holes are made with a tripod drill or bar, holes in the face in horizontal rows with a quarry bar, and in vertical or inclined rows with a gadder. A bar is better than a tripod drill for projecting rows of holes in a straight line, as it saves much time in moving.

Recently the jackhammer has replaced both bars and tripod drills in several quarries. It has a mechanical rotating device and may be run as a one-man drill. It uses hollow steel. It is run dry, and part of the exhaust passes down inside the drill and blows out the rock dust. On this account it is best operated by air, as steam condenses and forms a mud which is removed with some difficulty. When operated with steam it ceases to be a one-man drill, as a helper is required to pour water down the hole.

The jackhammer is operated as a hand drill, and is a great time-saver, as a few seconds only are required to change steel or to move to a new hole.

In one Alabama quarry jackhammers have now replaced all bar and tripod drills. In 1913 and 1914 tripod and bar drills were used, and in 1915 jackhammers took their places. The increased efficiency that results is shown in the following table:

*Drilling record of a quarry in which jackhammers superseded tripod and bar drills.*

Period.	Linear feet of drill hole cut per month.	Cost per linear foot.	Cutting rate in 10-hour day.	Cutting rate in 10 hours of actual cutting.	Time efficiency.		
					Proportion of time drilling.	Proportion of time lost in repairing.	Proportion of time lost in moving.
1913.....	<i>a</i> 11,981	\$0.0780	<i>Linear feet.</i> 99.7	<i>Linear feet.</i> 132.6	<i>Per cent.</i> 78.63	<i>Per cent.</i> 0.70	<i>Per cent.</i> 20.43
1914.....	<i>a</i> 8,699	.0682	106.0	135.0	79.00	1.09	19.91
1915.							
January.....	6,022	.0404	90.0	106.2	88.74	0.91	10.35
February.....	7,203	.0332	105.4	111.0	95.02	0.58	4.39
March.....	7,760	.0392	113.0	117.0	96.50	0.43	3.06

*a* Average of 12-month period.

The cost per linear foot in March, 1915, was just half as much as the average for 1913, and 40 per cent less than the average for 1914. This reduction in cost was due partly to the employment of one



instead of two men for each drill, and partly to the increased time efficiency indicated. In 1913 and 1914 the drills were operating on an average only 79 per cent of the time, whereas about 20 per cent was required for moving. In March, 1915, the jackhammers were actually at work 96.5 per cent of the time, and only 3 per cent was required for moving.

Other hollow-steel drills of similar type are now on the market, but are not in as general use as the jackhammer, and no figures concerning their efficiency were obtained.

A highly efficient bar drill employed by a Georgia marble company is used for both vertical and horizontal rows of holes in the quarry face. The bar is more than 12 feet long, and two drills are attached to it. In making a horizontal line of holes, each driller completes 6 feet of holes, and then the whole outfit is moved 12 feet, and the process is repeated. In drilling the face, two vertical rows of holes are projected simultaneously, one drill operating on each row. At the completion of each pair of holes the bar is moved up to the proper position for the next pair. Much time is thus saved as it takes no longer to adjust the bar for two drills than for one.

With the exception of the jackhammer, most drills work successfully with either steam or compressed air. If electric power is available, drills of special design may each be operated by a portable motor-driven pulsator. This method is especially advantageous if drilling operations are carried on at a considerable distance from the power plant, as power can be transmitted with greater ease by wire than by pipe lines or hose, and with less loss of power through leakage, friction, or heat radiation.

#### DRILLING IN RELATION TO CHANNELING.

The separation of rock masses is ordinarily less expensive per square foot by drilling and wedging than by channeling. The former method can be employed, however, only where a channel cut allows freedom of motion of the block in the direction in which the wedges tend to force it. Thus a certain amount of channeling is unavoidable. If there is a decided rift in the marble, drill holes may be placed much farther apart when splitting is being carried in the direction of the rift than when the splitting is carried across the rift. On this account when the wall cuts are made, if the direction of channeling is not influenced by bedding or unsoundness, one set of channel cuts should be made in a direction at right angles to the rift, and the cross breaks made by drilling and wedging. Advantage is thus taken of the rift direction to reduce the number of drill holes. As regards "liver rock," a term applied to a marble that has no rift, the necessity for placing drill holes close together and the difficulty of obtaining

a uniform break if the plug-and-feather method is employed, may render it advisable to channel in both vertical directions and to wedge only on the floor.

#### ARRANGEMENT AND SPACING OF HOLES FOR CROSS BREAKS.

If the rock is uniform and sound drill holes may be spaced regularly in vertical holes. The spacing of drill holes varies from 4 inches to 2 feet, depending on the ease of splitting the marble. When the marble is unsound or lacks uniformity in color or texture, it may be necessary to make slanting cross breaks in order to avoid waste and to properly grade the product. Such breaks are shown in Plate V, B, p. 58.

#### DRILL HOLES FOR FLOOR BREAKS.

When channeling is done in one direction only, except for wall cuts, the long masses of marble are broken loose from the floor before cross breaks are made. Occasionally a floor cut is made with a channeling machine, but almost invariably they are made by drilling and wedging. As with holes for cross breaks, the spacing of the holes is governed by the ease with which the marble splits. A common practice is to make alternate holes shallow and the intervening holes the full depth of the break desired. The depth of each hole is marked on the rock surface for the guidance of the worker in selecting wedges. The holes may be parallel with each other or may radiate in fanlike form.

#### PROPER SIZE FOR DRILL HOLES.

Drill holes should be as small size as possible without detracting from wedging efficiency. Most drill holes are made  $1\frac{1}{2}$  to  $1\frac{3}{4}$  inches in diameter. It has lately been demonstrated that the diameter of drill holes can be greatly reduced without interfering with the successful operation of wedges. J. P. McCluskey, quarry superintendent of the Alabama Marble Co., found that on an average  $4\frac{1}{2}$  minutes was required to drill a  $1\frac{3}{8}$ -inch hole 2 feet deep. The time required to drill a  $1\frac{1}{8}$ -inch hole 2 feet deep was only  $2\frac{1}{2}$  minutes. All drill holes in the quarry of the company mentioned are now made  $1\frac{1}{8}$  inches in diameter. The change from  $1\frac{3}{8}$  to  $1\frac{1}{8}$  inches has resulted in a reduction of 44 per cent in the time consumed per linear foot of drilling.

In order to wedge successfully in holes of such small diameter, wedges of special design are used. They are described subsequently under the title "Wedging."

#### ADVANTAGE OF REAMING DRILL HOLES.

Acting on a suggestion made by the writer, a certain marble company tried the experiment of using a reamer to cut grooves on the opposite sides of drill holes, thereby assisting the splitting so materially that

fewer drill holes were required. The reamer was made in the form of a drill with wings at the sides sufficient to cut grooves about one-fourth of an inch deep on opposite sides of the drill hole. It could be used only in a reciprocating drill. When the hole was completed the drill bar was removed and the reamer put in its place. The rotating device was thrown out of gear, and by means of a bar through which the square shank of the reamer passed, the latter was held in proper position to cut the grooves exactly in line with the desired direction of splitting.

It was found that a straight break could be made with drill holes at least twice as far apart as when no reamer was employed. However, no reamers could be found that would bear the work required of them. The projections invariably broke off after short service. Until this mechanical difficulty has been overcome, the method is not considered to be practical.

#### SHARPENING AND TEMPERING OF DRILLS.

Economy in drilling depends to a great extent on the ability of the blacksmith to properly sharpen and temper drills. If the drills are too hard they will chip easily, and if too soft they dull rapidly. Uniformity in the size of drills is also important. Much annoyance and loss of time will probably result from an endeavor to continue a drill hole with a drill slightly larger than the one employed to drill the first part of the hole.

#### WEDGING.

##### TYPES OF WEDGES EMPLOYED.

In some quarries wedges that reach only a short distance into the drill holes are employed. As the entire strain is near the rock surface heavy sledging is necessary, and uneven fractures may result. The method is not effective.

The long wedges commonly used represent a marked improvement. Iron plates or feathers are attached to their extremities with wire in order to hold them in proper position while being inserted into the drill holes. When the wedges are driven the strain is thus exerted at points near the bottoms of the drill holes. If alternate holes are made half depth the strain is more evenly distributed.

It has been mentioned in a previous paragraph that one quarry company reduced the size of drill holes to  $1\frac{1}{8}$  inches. Ordinary wedges reduced to fit such holes were not strong enough to withstand the heavy sledging required. Consequently a wedge was designed that gave effective service with lighter blows.

The type that has proven highly successful is shown in figure 13. The feathers are 3 feet long and the plug 3 feet 9 inches, the additional



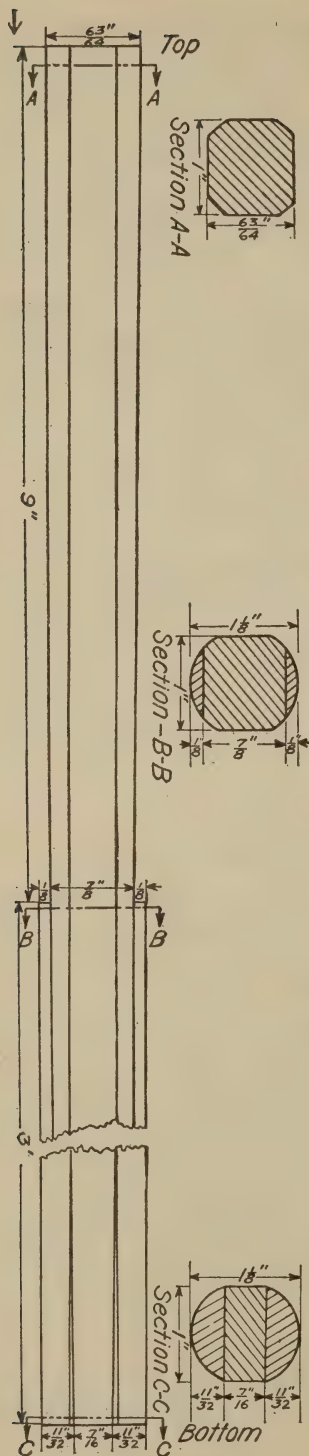


FIGURE 13.—Details of wedge that is effective in a small drill hole.

9 inches being required for driving. The plug tapers gradually from  $\frac{6\frac{3}{4}}{16}$  inch in diameter at the top to  $\frac{7}{16}$  inch at the bottom. The feathers are curved on one surface to fit the drill hole, and the flat surface is perfectly straight and gives a uniform taper from one end to the other. They are made true to form in a swage. The important feature of the design is that the diameter B-B at the top of the feathers is exactly the same as the diameter C-C at the bottom,  $1\frac{1}{8}$  inches. As a consequence when the plug and the feathers are inserted into the drill hole in the position shown in the figure, one side of each feather is in contact with the plug, and the other side with the wall of the drill hole throughout its entire length. When the plug is driven the feathers are forced apart a uniform distance at every point, and as a result the pressure exerted is uniformly distributed over their full length. Straight and even fractures result, and are obtained with much lighter sledging than by any other method yet devised.

#### UNIFORM STRAIN DESIRED.

It is important that in driving wedges there should be equal strain on all of them. If wedges near one end of a long mass of marble are driven much harder than those near the other end, an irregular cross break may result. In other words an artificial strain break may be produced, especially in long breaks in marble that has no rift. Wedging such marble should never be unduly hastened. A more uniform break will result by giving the rock time to fracture gradually than by forcing it. Uniformity of strain and moderation in the rate of fracturing may both be best accomplished by em-

playing one man only on a single floor break. Different men strike blows with different degrees of force, and thus although one man may drive a wedge with the same number of blows as another, one may exert a much greater strain than the other. Also when two or more men are employed, there is a tendency to force the break at a too rapid rate.

If short breaks are made, or if the marble has a decided rift parallel with the direction in which fracture is desired, more latitude may be allowed. In some quarries several men work together in driving wedges, a leader giving the word so that all strike at once. A gang may, with practice, work together in this manner and produce a uniform strain. If the rapid splitting has no ill effect the method is justified.

#### EFFECT OF RIFT ON DRILLING AND WEDGING.

It has already been stated that the process of splitting is greatly facilitated by rift. If there is an exceptionally pronounced rift in a horizontal direction, it may be possible to make a floor break double the width of an ordinary marble block. This method is employed by a Maryland company. After the wall cuts have been completed, channel cuts are made 12 feet apart and in one direction only. A floor break is made the full 12 feet wide, and then the mass of marble is split lengthwise by drilling and wedging. Thus, in one direction, the rock is intersected by alternate channel cuts and rows of drill holes, and all the cross breaks are made by wedging. As the cost of drilling and wedging is less than that of channeling per square foot of surface produced, a considerable saving is effected.

In most marble there is a decided rift parallel with the bedding. Therefore, if the bedding dips at a steep angle, the rift may be inclined in like manner. If the rift is inclined and the quarry floor level, the direction in which drill holes are projected for floor breaks is a matter of considerable importance. In a Colorado quarry in which the floor is level and the rift steeply inclined the channel cuts are made parallel with the strike of the rock. The allowance that should be made for the influence of rift on the process of wedging under such conditions is shown in figure 14. When the row of key blocks has been removed holes may be drilled in the direction shown by the arrow *a* in the

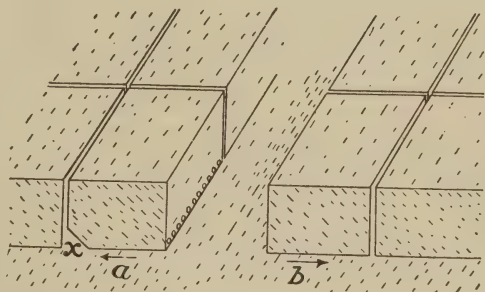


FIGURE 14.—Influence of rift on bottom breaks. Holes drilled in direction *a*, if too shallow, will result in break at *x* and injure blocks; holes drilled in direction *b* will result in a straighter break.

figure. When the holes are wedged there is a tendency for the break to leave the plane of the drill holes and slant upward on the rift, thus removing a corner of the block, as at *x*. When the holes are drilled in the opposite direction, shown by the arrow *b*, if the channel cut is not continued lower than the plane of the drill holes, the break will be straight, as it will not run down below the bottom of the channel cut. As a consequence, when the holes are drilled in the direction *a* they must be made deeper than when drilled in the opposite direction. It is apparent that, to avoid waste by broken corners and to reduce the expense of drilling, the row of key blocks should be taken out as near as possible to the left side of the quarry as viewed in the figure in order that most of the drilling may be done in the direction *b*.

In cutting out masses of marble with the wire saw in this quarry it was difficult to maintain a uniform quarry floor. The saw cuts were projected downward 18 or 20 feet. In making the floor breaks in the direction *a* (fig. 14) there was a tendency for the break to pass upward and remove a corner of the block, and in making the floor break in the opposite direction the break would slant downward on the rift and remove a corner of a block on the floor beneath.

#### REMOVAL OF KEY BLOCKS.

Various methods are employed for removal of key blocks. As in many other quarry operations, the method is controlled to some extent by the conditions and to some extent by the skill and experience of the foreman. The key block may be removed in fragments as waste, it may be divided into a number of small blocks, or it may be removed in its entirety.

#### REMOVAL OF KEY BLOCKS AS WASTE.

If a band or mass of poorer material traverses the quarry, the key blocks may be located in it. The mass may be blasted into fragments and removed as waste. This method is employed by a North Carolina company. A narrow band of flinty inferior material passes through the middle of the quarry. The key blocks, consisting of this flinty material, are removed by blasting, and no unnecessary loss ensues.

However, some quarry experts think that even inferior rock can be removed more cheaply in large masses with the derrick than by breaking it and removing it in small fragments. In areas beyond the reach of the derrick the latter method is to be preferred.

Even if no such inferior bands or masses occur, some quarrymen blast and remove the first key block as waste, believing this method to be the most rapid and convenient. Unless the marble thus quarried is of low grade, such economy of time will not justify the waste of material.



## MAKING THE FLOOR BREAK.

When the key block is to be preserved, the first step after channeling has been completed is to make the floor break for the first block. A common method is to insert a slanting iron plate in the bottom of the channel cut and to place the point of a wedge between it and the key block. When the wedge is driven, the entire strain is exerted at the bottom of the block. A series of such wedges may be placed close together. If the floor is intersected by other parallel channel cuts, it is necessary to place wedges in those immediately behind the point where wedging is done, to insure that the fracture will take place where desired.

A method now rarely used is to charge an iron tube with blasting powder, tamping it hard above the charge and plugging it lightly below. The end of the pipe containing the charge is placed in the bottom of the channel cut and a strain put on the block by driving wedges. When the charge is fired, a clean break is said to result from the shock. A small charge placed in a hole drilled in the center of the block the exact depth of the channel cuts will act in much the same manner. Such a hole may, however, constitute a serious defect in the block.

After the first block has been removed, horizontal holes may be drilled and the next block broken loose by wedging in the ordinary manner. When the working space is too confined for a bar drill, a jackhammer may be used. When the bench has an open side, the first block may be drilled by placing a drill bar in a waste box and suspending it in proper position with the derrick.

A special method is employed in a Vermont quarry, where operations are conducted on a floor slanting about  $45^{\circ}$ . If key blocks were drilled and wedged in the ordinary manner, there would be great danger of the blocks sliding down upon the men the moment they were broken loose. To overcome this danger, a single drill hole is projected at the center of the floor line, and a light powder charge is exploded in it. The charge is so small that it makes the floor break without otherwise shattering the block.

## HOISTING OUT KEY BLOCKS.

A method that is too wasteful to be recommended is to quarter the first key block with the channeling machine and remove it in sections. Removal in four pieces is undoubtedly easier than in one. There are two serious objections, however—first, the additional expense of making the two extra channel cuts, and, second, the decreased value of the material in small blocks. It is probable that the marble in four pieces is worth less than half as much as the intact block.

There are now in common use three methods of removing the first key block in its entirety. The first of these is by the use of the Lewis pin, which is adapted only to strong rock. A hole several inches deep is drilled into the middle of the block. A bar with an eye in the top is placed in the hole with a wedge at each side of it, as shown in figure 15. The bar is thicker at the bottom than at the top, so that when pulled upward it tends to tighten on the wedges, when the block may be lifted out with the derrick hoist.

A second method which may also be employed in strong rock is the use of grabhooks. Small pieces may have to be broken from the corners of adjoining blocks in order that the holes may be drilled properly and that there may be room for the hooks.

A third method is employed if the beds are weak. Chain loops or cables are thrown over the block from opposite sides. They are placed near the bottom of the block and are drawn tight and the block is lifted out with the derrick hoist.

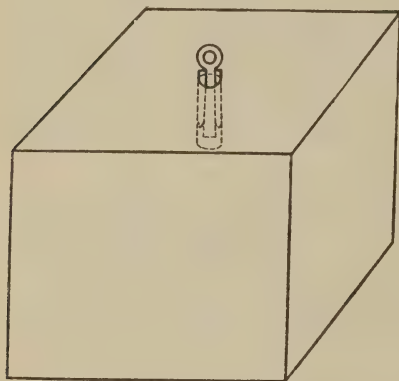


FIGURE 15.—Lewis pin used in hoisting key blocks.

The block is then raised several inches with the derrick and blocked up ready for final cable attachment.

The employment of the derrick to assist the bar gang in turning down blocks may result in serious risks to those employed. After the first fracture has been made a block is by no means free. The interlocking of irregularities on the surface may require considerable strain before the block will move. When the derrick is employed there is grave danger of breaking a boom or cable with consequent danger of serious accident.

#### HOISTING.

##### TURNING DOWN BLOCKS.

Blocks are usually turned down with a quarry bar before the hoist cable is attached. When monoclinic blocks are excavated it is difficult to turn them down. In a Vermont quarry the tip of the block is raised with a bar and a cablesling placed beneath it. The

##### CABLE ATTACHMENT.

The hoist cable may be attached to blocks by grabhooks or chain or cable slings. Grabhooks are employed only with rock that is hard and coherent. The two holes for the hooks are made on opposite sides of the block a few inches from the top. A mistake is

sometimes made in drilling the grabhook holes too deep. The chief strain then comes, not at the tips of the hooks, but on the curved parts that are in contact with the upper edge of the block. As a consequence a corner of the block may chip off and allow the whole mass to fall. The holes should be sufficiently deep to allow a firm grip of the rock but the chief pressure should fall on the tip of the hook in the bottom of the hole. Moreover, great care must be taken in hoisting by this method. The rock should be carefully balanced, as a partial rotation may cause the hooks to slip. Under no conditions whatever should a quarry workman occupy a position beneath a block that is being lifted with grabhooks.

A safer method of attachment is to pass chains under the block and completely around it. Blocks are held much more securely by this method and there is little danger of a mass of rock falling on account of a weak bedding plane or fracture. A chain is, however, rather uncertain in its strength, and detection of weak links is somewhat difficult. When a chain does break the accident is usually quite unexpected. A wide margin of safety and frequent examination are necessary.

The best method of cable attachment is by means of cable slings (Pl. VI, A). Such slings are quickly handled and the use of two slings renders balancing of the block comparatively easy. Defects in steel cables are easily recognized. A realization that slings are both safe and convenient has led to their adoption in many quarries.

#### EFFICIENT HANDLING OF MATERIAL.

All unnecessary handling of material should be avoided. Blocks should be hoisted from the quarry and loaded on cars at a single operation if possible. If circumstances will not permit this, they should be placed in a convenient position for future loading. At most quarries hoisting and loading are done in a fairly efficient manner.

However, more efficient methods of handling waste material should be devised. In many quarries there is great loss of time and power in rehandling waste blocks. At one quarry waste blocks were hoisted from a point near the surface at one side of an excavation, thrown to the bottom of a 50-foot pit, picked up with a second derrick, and deposited on cars. Cars were situated within easy reach of the first derrick, but the placing of the blocks on them necessitated swinging the blocks over channeling machines. Usually such conflict of operations may be avoided by careful and judicious plans for quarry development. Every additional operation in handling waste adds to the cost per cubic foot, and every increase in such cost cuts down the margin of profit on the finished product.



## TUNNELING.

## DEFINITION OF TUNNEL.

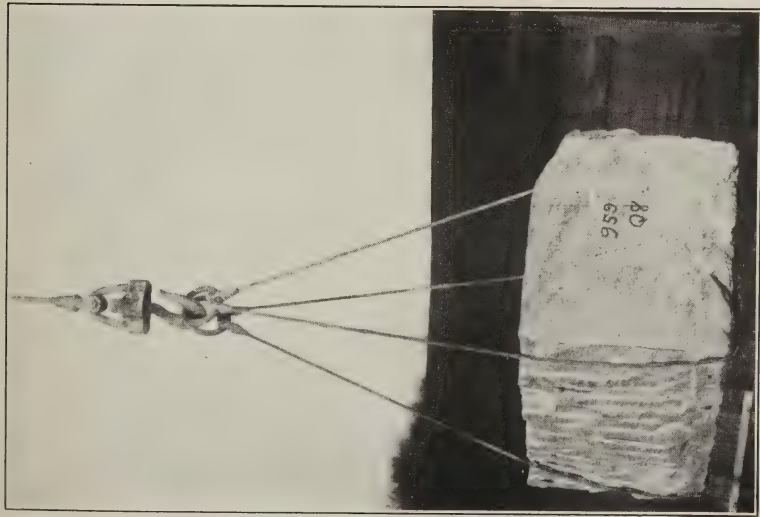
In quarrying the term "tunnel" is applied to a subterranean working, level or inclined, having a roof of undisturbed rock. The term is used in contrast with the "open-pit" quarry in which the opening is the full size of the excavation, and with the "undercut" quarry, the walls of which slant so as to make the floor space wider with increasing depth.

## GENERAL CONSIDERATIONS.

Although open-pit quarrying is the more common type, there are conditions under which tunnel methods are to be preferred. A tunnel affords a means of quarrying out desirable beds without the removal of heavy stripping. However, there are certain difficulties and dangers that must be carefully considered. Among them may be mentioned the danger of roof falls, the expense of artificial light and ventilation, and the necessity of additional handling of quarry material. A necessary condition for successful tunneling is the presence of a strong roof. In regions where faulting or jointing are pronounced the danger from roof falls may be so great that tunnel methods are impracticable.

## OPENING TUNNELS ACROSS BEDS.

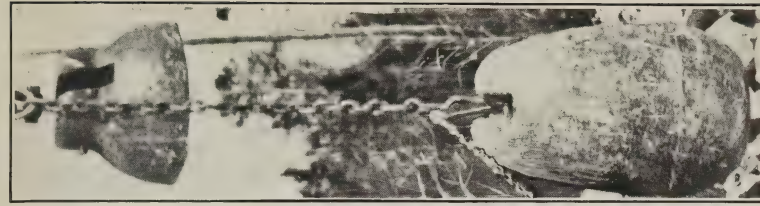
Procedure to be followed in tunneling a deposit in which there are no open bedding planes or when the tunnel roof is to cross the beds may be illustrated by describing the method used in a Vermont quarry. A preliminary opening 6 to 7 feet high is made by channeling and blasting. As the rock thus removed is waste, the opening is, if possible, made in inferior material. Determination is made that the thickness of sound rock above the tunnel is such that there is no danger of roof collapse. After the proper position for the ceiling has been chosen a channeling machine is used to cut a channel about 7 feet deep, starting about 3 feet from the floor and slanting downward to meet the floor line. A row of horizontal drill holes is then made at the roof and another row is made parallel with the floor of this preliminary tunnel. In addition vertical rows of holes are driven at intervals of about 7 feet. The relative positions of channel cuts and drill holes are shown in figure 16. The lower wedge-shaped mass of rock *x* in the figure is dislodged by blasting in the drill holes below the channel cut. The upper overhanging ledge *y* is then broken down by discharging blasts in the holes above the channel cut. Black blasting powder is used and all charges are exploded by fuse. Such blasting is probably less effective than a simultaneous discharge of



4. CABLE SLING FOR HOISTING BLOCKS OF STONE.



B, UNDERCUTTING PARALLEL WITH STEEPLY INCLINED BEDS IN QUARRY AT WEST STOCKBRIDGE, MASS.



C. A BALL BREAKER USED IN QUARRY AT GANTTS QUARRY, ALA.





blasts by means of electric detonators. However, if the tunnel is driven in tight beds or if it crosses the beds the simultaneous discharge of a number of blasts would undoubtedly shatter and thus destroy marble adjacent to the tunnel.

When, by repeated blasting, the mass of rock 6 or 7 feet in depth has been shattered across the entire width of the tunnel, the next step is to remove the waste material. Large pans holding 2 to 6 tons each of rock are used. Each pan is shaped like a sugar scoop and is handled by means of a cable fastened to the back of the pan, passing to a second attachment at the front, and then over a sheave in the tunnel roof near the edge of the excavation. Rock fragments are loaded in by hand and the pan is hauled to the edge of the shelf. The cable is then released from the front of the pan, and further hauling causes the back of the pan to be raised, allowing the contents to slide out.

If the tunnel floor slants downward from front to back, the empty pan is returned on rollers, whereas if the floor is level or inclined upward the pan is hauled back by means of a cable attached to the back and passed around a sheave at the extremity of the tunnel. The rock fragments are again loaded into pans at the base of the shelf, hauled by trolley or cable cars to the main hoist, elevated to the surface, and loaded directly on railroad cars with the derrick.

The process of channeling, drilling, blasting, and removal of fragments is repeated over and over until the preliminary tunnel has been projected far enough to give sufficient floor space for economical operations. Plate VII, A, shows the appearance of a tunnel in one of the Vermont quarries.

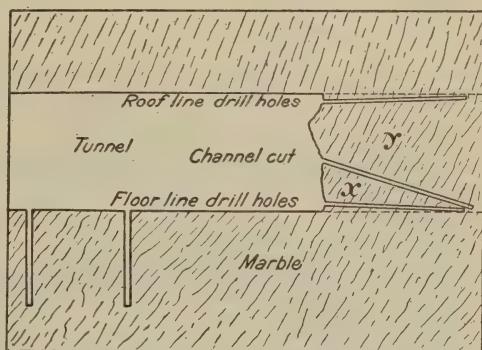


FIGURE 16.—Method of driving a tunnel in marble.

#### METHOD OF OPENING TUNNELS PARALLEL WITH OPEN BEDS.

In an Alabama quarry tunnels are driven parallel with the beds, which dip about  $33^\circ$ . The tunnel floor is made coincident with a band of soft mica schist which occupies a position between two marble beds. The soft band is thus utilized to take the place of the channel cut as described in the preceding method. One expensive operation is thus saved. Drill holes are projected in a row parallel with the roof, and vertical rows of holes pass from the

roof to the floor at intervals. Black blasting powder shots are discharged in simultaneous groups by an electric firing machine. The presence of the band of mica schist at the floor acts as a cushion and prevents the shock from shattering the good marble below. On this account, the more effective simultaneous blasting may be employed.

In removing the material, loaded buckets holding 4 cubic yards are hauled to the edge of the excavation but are not dumped. By means of a derrick at the mouth of the tunnel they are transferred to a position from which they may be hoisted with the main derrick and dumped into railroad cars.

#### METHOD OF QUARRYING ON TUNNEL FLOOR.

When a preliminary tunnel of sufficient size has been completed, channeling machines may be set up on the floor and operated as in an open quarry. Electricity is the most convenient power for use in tunnels. Air-driven machines are better than steam-driven, as the space for operation is usually confined. The channelers may work on a level or an inclined floor. At West Rutland, Vt., the upper levels are quarried on a horizontal floor and the lower ones on an inclined floor. In an Alabama quarry the channel floor is inclined and the channel cuts intersect the slanting floor diagonally.

#### ROOF SUPPORTS.

In a West Rutland, Vt., quarry the tunnel roofs are supported by pillars of marble each 20 feet square left at various intervals. The spacing of the pillars is governed by the evident security of the roof. If the beds are heavy and seams absent, the spacing may attain a maximum of 100 feet. In sections where the beds are seemingly less secure the pillars are spaced at intervals of 60 to 80 feet. In early days if the roof slanted the pillars were inclined to form right angles with the roof. Inclined pillars, however, occupy more space, and they form more serious obstacles to the operation of derricks than vertical pillars. Of late years all the pillars are made vertical though the roof may be inclined.

In an Alabama quarry a wall running in the direction of the dip is left, rather than a series of pillars. This plan was adopted after due consideration of the possibility of pillars slipping on a soft, inclined, interbedded mica schist.

Figure 17 illustrates the present and proposed future arrangement of pillars in a Colorado quarry. The tendency to stagger alternate rows is noteworthy.



A. A TUNNEL IN A VERMONT MARBLE QUARRY.



B. GANTRY CRANE EMPLOYED AT QUARRY AT ASHLEY FALLS, MASS.



C. MANNER IN WHICH OPEN SEAMS PITCH INTO THE CORNER OF A QUARRY WHERE THE QUARRY WALL MEETS THE STRIKE OBLIQUELY.





## TRANSPORTATION OF MATERIAL IN TUNNELS.

Blocks of marble or boxes of waste material may be hauled out of tunnels by means of cables. Quarry cars operating on tracks are more efficient. One company operates both cable cars and an electric trolley. If an efficient means of transportation is not provided, great loss of time will result.

## LIGHTING AND VENTILATION OF TUNNELS.

In order to promote safety and efficiency in tunnels, adequate lighting is necessary. Either arc or powerful incandescent lights are suitable. If blasting is employed, the harmful gases should be

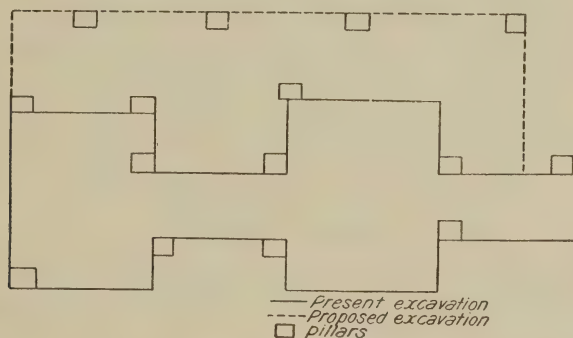


FIGURE 17.—Present and proposed arrangement of pillars in a Colorado quarry.

removed. In the ventilation of a Vermont quarry large wooden conduits through which the impure air is drawn by powerful fans are used.

## UNDERCUTTING.

A modification of the tunnel method consists in enlarging the quarry floor by an outward inclination of the wall cuts. The process is simple, requiring no additional equipment and no expensive preliminary operation. A wide floor space is obtained with a minimum of stripping, and with moderate extension no supporting pillars are present to obstruct quarry operations. There are, however, certain disadvantages. In tunneling, the projection of a preliminary opening is costly and produces only waste rock, but when once completed the subsequent operations of channeling and drilling are carried on with almost the same facility as in an open quarry. In undercutting, however, every wall cut is slanting. Channeling at an angle is slow and relatively expensive. Also the blocks of the outer row are angular in shape, resulting in waste.

In extensive undercutting, the danger from overhanging rock may be averted by leaving wing supports of marble at intervals. Under-

cutting is employed successfully in many of the Georgia marble quarries. It is to be recommended if the rock is sound, and if tunnel methods are too expensive or for other reasons seem impracticable.

In a Massachusetts quarry the walls are undercut parallel with the steeply inclined beds as illustrated in Plate VI, *B*. By this means the desirable beds are followed and the floor space gradually increased. The blocks of the first row are removed as keys and subsequent channels are cut at right angles to the strike.

#### COMBINED TUNNEL AND UNDERCUT.

If a band of marble is vertical in attitude or inclined at a steep angle and not exceeding 150 or 200 feet in thickness, a safe and economical method of extraction is to project a narrow tunnel and gradually widen it by undercutting. If through weathering the marble has been altered to a considerable depth, it may be possible to project the preliminary tunnel in inferior rock. Undercutting will gradually widen the floor, and the steep walls thus formed are less liable to scale than a flat roof. Undercutting is not to be recommended, except when the lateral walls are to be permanent, for as a rule, it is not practicable to further extend undercut walls. When the lateral limits of the desirable material have been reached, the angle of inclination of the wall cuts may be modified to conform with the dip of the marble beds.

#### DRAINAGE.

There is great variation in the quantity of water encountered in quarries. In some quarries heavy springs are encountered which require constant pumping; in others surface drainage only must be removed. Steam or motor driven reciprocating pumps are most commonly employed, though motor-driven centrifugal pumps have been used successfully, and are highly recommended by a number of quarry operators.

#### GENERAL CONSIDERATIONS IN QUARRYING.

Whenever possible, it is wise to conduct quarry operations on a wide floor space. In general, operations on a large scale are more economical than those on a small scale. A large floor space also allows greater choice of material. The encountering of local unsoundness may temporarily suspend production in a small quarry, whereas in a larger one the unsound rock may be avoided for the time being and removed during a dull season.

Greater efficiency in quarry operations may be attained by employing night shifts. By this means the output can be almost doubled without any increase in the power plant or quarry equip-



ment over that required for the day shift only. The work should be so arranged that the night shift is relieved of those operations that on account of poor light, or for other reasons, are not performed to advantage in the night. Machine work is the most satisfactory of all operations for night shifts, and consequently, the day shift should endeavor to leave the machines in the best possible condition for continuous operation.

All quarries should be provided with safe and efficient stairs or ladders.<sup>a</sup> Much time is lost when men are obliged to pick their way over steep and rough rock surfaces or climb down hastily improvised steps or ladders.

Removal of mud and rock fragments from the quarry floor is an operation that should not be neglected. An accumulation of débris impedes quarry operations. The presence of a large heap of rubbish in one part of a quarry may discourage the proper performance of a task and lead to the substitution of a less-efficient method. All rubbish should be cleaned up systematically and frequently, and the entire floor space kept in a condition favoring a free and intelligent performance of all subsequent tasks.

#### SAFETY FIRST.

"Safety first" should be the motto in planning and erecting all quarry equipment and in the conduct of all operations.<sup>b</sup> Many operators fail to realize that safety devices contribute to efficiency. They consider that much of the money spent in safeguarding machinery and the time employed in making examination for possible dangers is entirely lost. The matter of safety should be viewed in a different light. Men become greater producers if they feel that all reasonable means are employed to safeguard their lives and health. Much time is lost if men feel that they must constantly be on the lookout for some overhead dangers. Moreover, accidents deprive the quarry of the services of skilled men and frequently burden the proprietor with a load of expense.

#### HOISTING EQUIPMENT.

##### TYPES OF DERRICKS.

Various types of derricks are employed in marble quarries. Both the mast and the boom may be of wood, the mast may be of wood and the boom of steel, or both may be of steel. Spliced wooden derricks having mast and boom each in four pieces are used in some regions. They are strongly supported with iron bars and turnbuckles. They are easy to transport and set up. Those in common use

<sup>a</sup> See Bowles, Oliver, *Safety in stone quarrying*: Tech. Paper 111, Bureau of Mines, 1915, pp. 15-16.

<sup>b</sup> See Bowles, Oliver, *Op. cit.*, pp. 12-36.

have a lifting capacity of 15 to 18 tons. All derricks should be painted to protect them from the weather.

#### USE OF GANTRY CRANES.

A Massachusetts marble company uses a traveling gantry crane in place of a derrick, as illustrated in Plate VII, *B*. It has an 80-foot span and a lifting capacity of 25 tons. The entire crane travels back and forth on tracks placed at each side of the excavation. It is readily accessible to all parts of the quarry, and blocks once lifted are transported directly to the stock pile without further handling. One disadvantage is that the use of the gantry crane limits the width of the quarry. Furthermore, it is not to be recommended for quarries in the development of which considerable hoisting will be required, as it necessitates that all lifting be done with one hoist. If derricks are employed several may be placed in convenient locations and operated simultaneously.

#### DERRICK GUYS.

All derricks should be strongly supported. For 20-ton derricks eight iron guys of  $1\frac{1}{4}$ -inch diameter are necessary. For derricks with 90 to 100 foot masts and with a maximum lifting capacity of 35 tons the guys should be  $1\frac{1}{4}$ -inch steel. For locking guys, Crosby, Roebing, or Leschen clips are regarded as more reliable than splices.

A new arrangement of guys is now employed in some quarries in Vermont. Instead of 12 guys being spaced regularly, they are arranged in four sets of three each, the groups radiating at successive angles of  $90^\circ$  from the mast. An advantage lies in the fact that four large secure concrete piers may be constructed at a smaller cost than 12 small and possibly unsafe ones. Where several derricks are situated close together, it is often a problem to properly place guys and at the same time evenly distribute the strain. The arrangement mentioned overcomes this difficulty in a most satisfactory manner.

Where several derricks are situated near together, the top of one mast is occasionally anchored to the top of another. There is considerable risk in this method, as strains on different derricks may occur in the same direction simultaneously and thus multiply the strain on a single guy.

#### GUY ANCHORS.

Guy anchors are of various sorts. Where bedrock outcrops at the surface the safest method is to drill holes in the rock, insert a bar with a ring in the top, and fill up the hole with melted sulphur. Where considerable depth of soil surrounds the derrick, guys may be anchored to buried timbers. The trunks of large trees, preferably grow-

ing trees, are sure supports. It is advisable, however, to place a secondary anchor to support the tree.

A post supported by a pile of stones, as shown in Plate VIII, *A*, is used by some Tennessee quarrymen. The post should be nearly vertical. It should not be slanted backwards, as in that position it forms a lever tending to raise the soil or rock in front of it, and thus is not in the best position to withstand strain. Such posts should also be supplied with secondary anchors.

A concrete pier in which is embedded an angle-steel bar is a highly efficient support. An eyebolt may be passed through the bar and the slack of the guy taken up with a nut, as shown in Plate VIII, *B*.

#### HOISTS.

Steam hoists are the type most commonly employed in marble quarries, although compressed-air and electric hoists are used in several places. The type of hoist to be employed depends on the nature of the work. Deep quarries demand a higher cable velocity than shallow ones. In any case a very slow motion is not desirable. Much time is lost in quarries where men must wait while blocks are slowly raised to the surface, especially if rock excavation is being carried on near the point of elevation, as safety demands that men must move from their working places while the block is overhead. The power required for hoisting increases rapidly as the speed is increased, and a proper balance must therefore be maintained between the speed of hoisting and the power consumed for the operation.

In many quarries crude methods of sluing are still employed. Sluing is sometimes done by means of a rope attached to the block or by a mechanical sluing device operated by a handwheel. The sluing device that forms a part of all modern hoists is much more efficient, as it gives a more rapid motion than hand-operated devices and is controlled by the hoist engineer.

The spinning of blocks while suspended endangers the lives of employees and makes difficult the guidance of their course during ascent. Spinning may be avoided by using a nonspinning hoisting cable. The central strands of such a cable are twisted to the left and the outer strands are twisted to the right.

#### LOCATION AND SIZE OF DERRICKS.

The quarryman should have a clear and definite plan of quarry development before placing his derrick. The extension of a quarry excavation in a direction away from the derrick may soon bring it beyond the range of the boom. Thus the erection of a new derrick,



and possibly a second handling of material, may be necessary. In several quarries blocks are hauled long distances by cables before they reach a position from which they may be hoisted to the surface. Such preliminary haulage is injurious to cables, involves great loss of time, seriously interferes with other quarry operations, and is surrounded by many dangers. It may be advisable to install a car-haulage system or to replace or supplement a derrick by a new and larger one. The size of the derrick must also be governed by the extent and direction of quarry operations. A small derrick may give satisfactory service for a few months, after which it may have to be replaced by a larger one. It is usually more economical to erect a large and permanent derrick at first, provided the operator is reasonably certain that quarrying within its range is to be extensive.

#### HOIST SIGNALING.

In shallow quarries or in yard operations signaling by hand motion is almost universal. If the hoist engineer can not see the bottom of the quarry, hand-motion signals are sometimes relayed to a man in an intermediate position. Such a method is not to be recommended, as the repetition of signals involves increased danger of misunderstanding and also requires an additional man.

Types of mechanical signal apparatus in common use are electric bells, telephones, and bell pulls. Dry-cell electric bells are not considered reliable; they may fail to act at critical times. Electric bells operated by wet-cell batteries are more reliable if inspected at regular intervals. Telephone connections require inconvenient ear attachments. If noisy operations are conducted, words given by telephone may be easily misunderstood. The bell pull is considered to be the most convenient and reliable method. Both the apparatus and its operation are simple and there is little danger of misunderstanding signals.

#### SCABBLING.

Where the mill is situated close to the quarry, the process of scabbling may be omitted. If situated at a distance, or if the marble is to be sold in block form, the blocks should be scabbled to avoid transportation of waste material. The most common method of scabbling marble blocks is by manual labor with the scabbling pick. The wire saw is used successfully in some places. Hammer drills and wedges are used occasionally for the removal of the more prominent surface irregularities.



4. A GUY ANCHOR POST USED IN TENNESSEE.



B. AN ANGLE STEEL AND CONCRETE GUY ANCHOR USED IN  
A QUARRY AT GANTTS QUARRY, ALA.





**TRANSPORTATION OF QUARRIED ROCK TO THE MILL.****VARIOUS METHODS OF HAULAGE EMPLOYED.**

In some quarry regions mills are so favorably situated that short haulage only is required. In several eastern localities blocks are loaded directly upon the transfer cars by means of the quarry derrick. A number of companies use railroad cars and locomotives. Electric trolley lines are used successfully even on heavy grades. On such grades efficient and dependable brakes must be used. Steam tractors may be employed if roads are graded and firm. It is a matter of considerable expense to grade and keep in proper repair roads suitable for the heavy traffic incident to the use of tractors.

**TEAMS AND WAGONS.**

Teams and wagons are used in many places. Where quarries are situated on mountain sides and rock must be hauled down steep roads, the chain or shoe attached to the wheel to prevent too rapid descent is destructive to roads. Haulage by horse-drawn vehicles entails a slow rate of haulage, small loads, much repairing of roads, and heavy cost for maintaining horses. Hence, in many places, some form of cable-car transportation would seem to be more suitable than by teams and wagons. Where light grades are encountered, mules or horses may be employed to haul cars.

**CONNECTING QUARRIES WITH RAILROAD TRACKS.**

Many marble quarries are situated at short distances from railroads, and some form of conveyance is necessary to connect the quarry with the railroad track. Where the grade is light, a railroad siding is to be preferred. Where small quarry cable cars are used, it becomes necessary to erect a derrick at the junction point and re-handle all marble going out and all coal coming in. Where a siding is built, transportation may be conducted by means of standard railroad cars with no transshipping. Where a heavy grade is encountered, cable cars are necessary and transshipping may be unavoidable.

**CABLE CARS.**

Cable cars are used successfully in a number of regions. The conditions vary so greatly that no one particular form can be recommended. The system must be modified to suit the conditions.

One of the most complete cable-car systems in use in any American marble quarry is that in a Vermont quarry. Two cars are used, the empty car ascending while the loaded car descends. The hoist is operated by a 75-horsepower electric motor. A 1½-inch

steel cable passes from one car to the hoist where it makes six turns on a 12-foot drum, then passes out and is attached to the second car. The cable is guided by rollers. The track is three-fourths of a mile long. Over part of its course the grade is light, but at two points it is very steep. The cars pass at the center. Above this point a three-rail track is provided to prevent interference of the ascending and descending parts of the cable. Below the center the cable is always single, and in consequence a single track only is necessary. The maximum load is 40 tons, and 20 minutes is required for each trip. By means of an automatic needle indicator, which travels back and forth on a board, the hoist engineer can accurately follow the course of the loaded car. The rock is transferred from the cable cars to standard railroad cars for transportation to the various mills.

#### FACTORS CONTROLLING THE METHOD OF TRANSPORTATION.

Before a system of transportation for any particular quarry is adopted the conditions must be studied carefully. The most important considerations are probable output, size of material, number of times it must be handled, rate of haulage, distance, surface conformity of the country to be traversed, first cost of the system, cost of maintenance of equipment and roadway, and capital available for transportation purposes.

#### EQUIPMENT AND OPERATIONS IN MILLS AND SHOPS.

At least one-half the marble quarries of the United States have more or less completely equipped plants for sawing, polishing, carving, or otherwise preparing marble for structural uses. The operation and equipment in many such mills and shops were noted in connection with quarry observations. Marble mills and finishing plants belong to the manufacturing rather than to the producing phase of the marble industry, and consequently may not strictly be included in a discussion of the technology of marble quarrying. The brief discussion of marble-manufacturing plants presented in this section is justified by the direct interest that many quarrymen have in the manufacturing phase of the industry. Although certain facts relating to method and equipment are presented, for whatever immediate benefits they may suggest, no attempt is made to deal with the subject exhaustively at this time.

#### LOCATION OF PLANT.

The location of marble works is influenced greatly by water supply and availability of power. Aside from these considerations, although in many respects it is more convenient to erect the plant near the quarry, conditions may be such that this plan is unwise. If the

quarry is situated several miles from towns, in a region where roads are poor, it may be wiser to build the plant at some town, especially if other marble plants are centered there.

There are several arguments in favor of placing stone manufacturing plants at a common center rather than at the quarries. Although such an arrangement requires transportation of rock that is later wasted in saw cuts and coping, buyers can the more readily conduct their business if shops are centralized. Furthermore, it is easier to keep men near towns where schools and other public institutions are more convenient and more efficient than in comparatively unsettled regions. If shops are situated at a considerable distance from railway lines, sidings are often uneven and transportation over them is more destructive to thin-finished material than to uncut blocks.

### BUILDINGS.

Fireproof mills and shops are to be preferred. When a fire occurs the loss is great, because the heat is liable to destroy all the machinery. If buildings are not already of fireproof construction, a sprinkler system is to be recommended. One advantage of such a system is that water can do little damage. It may possibly cause swelling of the timbers, thus throwing the machinery out of alignment, but otherwise most of the equipment is of such a nature that water will not affect it.

The plan of shop and mill construction carried out by a Georgia marble company is worthy of remark. The company constructed with brick and concrete an absolutely fireproof section separating two units that were not fireproof. With such an arrangement fire can not paralyze production, because if one unit is destroyed the fireproof section prevents spreading of the fire, and operations may be continued in the remaining unit.

### HEATING AND VENTILATION.

In cold climates the necessity of preventing the freezing of water employed in the various processes, and also the need of keeping men comfortable while at work, demands that shops be closed structures. A system of heating and ventilation also becomes necessary. Coal stoves are employed in many mills and shops, but are not satisfactory, especially in large buildings. They require too much attention, increase the fire risk, and provide an unequal distribution of heat. In a Vermont mill the air, impelled by powerful fans, is passed through coils over a furnace, and conducted through overhead pipes, from which it is expelled downward at various points. With adequate renewal of circulating air from outside sources, heating and ventilation may be combined into one system. A Pennsylvania company uses the exhaust steam from its power plant to heat the mill and shop.



### POWER.

Direct water power is highly satisfactory. The cost of operation and maintenance of turbines is relatively low. An adequate water supply at all seasons of the year is desirable in order that continuous operation may not be interrupted through lack of power. At one Vermont mill two turbines on vertical shafts supply power to an overhead countershaft by means of bevel gears. The turbines are so arranged that one can be disconnected at night or at such other times as only partial power is required.

Steam is a common source of power for mills and shops. Where fuel is cheap steam is satisfactory.

Electricity, which may be developed by the company or purchased from power lines, is used in many places. Where an abundant supply and a good head of water may be obtained with a small outlay of capital, hydroelectric power may be cheaper than steam even though the cost of coal is low. The convenience in transmission of electricity has induced some companies to develop it by steam in places where there is not a sufficient water supply.

Where electricity is used, one motor may provide power for the entire mill. In other places the mill is divided into separate units driven by independent motors. One Tennessee company has one motor for each set of three saw gangs. One advantage of such an arrangement is that one part of the mill may be shut down for repairs to the motor while the remainder is in operation. Some mill men object to the use of electricity if the power is conveyed over transmission lines. To guard against burning out motors by overcharge due to lightning, a circuit breaker is necessary, and during a thunderstorm the circuit may be broken and all the machinery stopped. When saw gangs stop suddenly, sand packs in around the blades, and may make starting difficult or impossible.

### METHOD OF POWER TRANSMISSION.

For transmission from fly wheel to countershaft pulley two types of belts are employed—a broad belt of leather or fabric and a rope belt. The latter has the advantage of low first cost and is easily tightened, the tightening pulley being applied to a single turn only of the rope.

Direct water power is commonly transmitted by gears. Wooden cogs in one of the intermeshing wheels and iron cogs in the other have been found a very satisfactory combination. A skillful carpenter can easily replace defective wooden cogs, and such cogs, if properly made, are durable.

## PLAN OF MILL, SHOP, AND YARD.

Stone is heavy and excessive handling adds greatly to the cost of the finished product. Consequently, marble producers should aim to arrange mill, shops, and yard in a manner that permits minimum handling. Where sawing and finishing are both done, it is a common practice to place mill and shop parallel, 30 to 60 feet apart, and to have an overhead traveling crane between them. Where sawing and

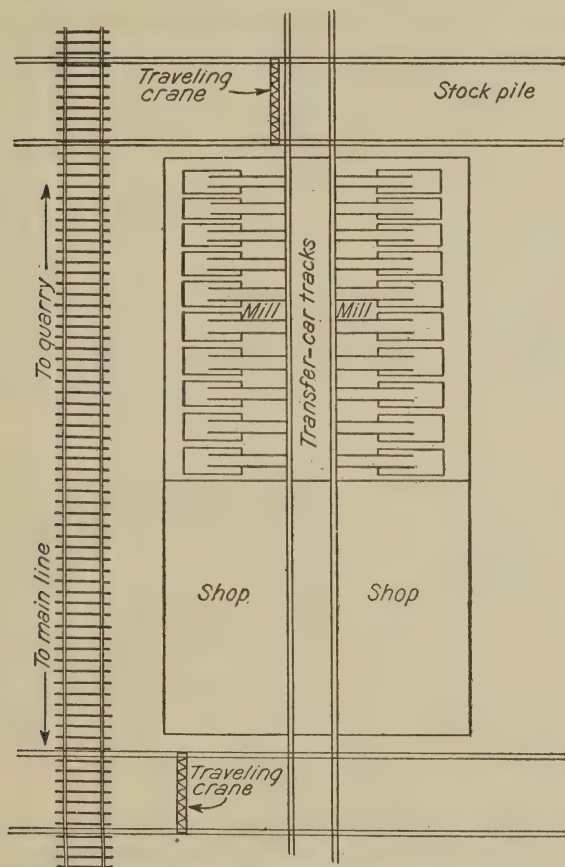


FIGURE 18.—Track arrangement at a Vermont marble mill.

finishing are conducted on a large scale, the plan of a Vermont mill is efficient. The essentials of the plan are shown in figure 18. One traveling crane is employed to unload blocks from cars on their arrival at the mill and to pile them with the stock or to load them on to the transfer cars. A small locomotive crane is employed to haul the transfer cars through the mill, where a track passes down the center with gangs on either side. Beyond the mill is the finishing

plant and at the end of it another smaller traveling crane for loading the finished stock on railroad cars.

A convenient plan for a small plant having a mill but no shop is that of a Maryland plant, as shown in figure 19. Gang cars are used but no transfer cars. Cars are loaded by either of two derricks situated at opposite sides of the mill. Railway tracks pass at both sides of the mill for bringing marble blocks, sand, or fuel, and for taking

away finished stock. Both rough and finished stock are piled within reach of one or other of the derricks.

MILL.

SAW GANGS.

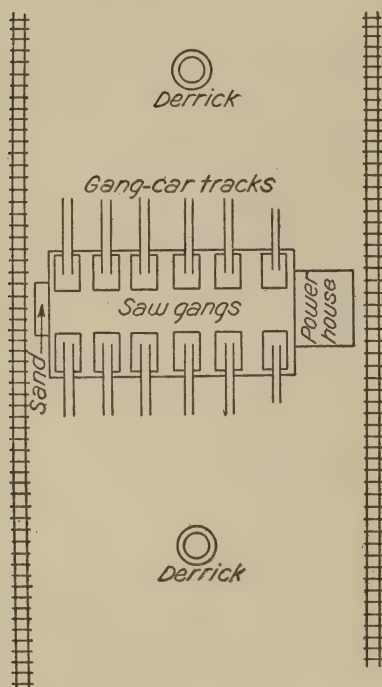


FIGURE 19.—Plan of mill and yard of a Maryland marble company.

Saw gangs constructed a number of years ago have wooden frames. Such frames are not to be recommended, as frequent wetting and drying cause them to decay rapidly. Modern gangs have steel frames which are more rigid and durable. An efficient frame devised by a Georgia company consists of iron tubes 8 or 10 inches in diameter, which were set up and filled with concrete. Such frames are constructed cheaply.

All hanger bearings for the gangs should be so capped that they are perfectly sand proof.

#### ABRASIVES USED IN SAWING.

The most common abrasive used in sawing is clean sharp sand. Many millmen use crushed steel or fine steel shot, either with water alone or with sand and water. The resulting increase in the rate of sawing is variously estimated by different operators at 20 to 50 per cent. As all millmen know, if the saws are fed too fast the blade bend and cut uneven surfaces, which must later be rubbed for a long time to be made uniform. Although it is possible to crowd saws to excess when crushed steel is used as an abrasive, the liability to crowding is less than when sand is used alone.

Certain rules must be followed in the use of crushed steel as an abrasive. It should never be used with rock that contains pores large



enough to allow grains of steel to become lodged therein. The same is true of marbles containing soft minerals, as mica. If the grains of steel become lodged in the pores or in the soft minerals, they may later cause scratches when the rock is under the buffer. Also porous marbles, especially those that exhibit a marked permeability, are apt to stain with the iron rust formed from the steel abrasive. A steel abrasive may be used successfully with marbles having small pores and no streaks or bands of relatively soft minerals.

It is well known that the presence of carbon dioxide greatly increases the tendency of iron to rust, and its complete absence renders the process of oxidation (rusting) extremely slow. On this account many millmen mix lime with the steel abrasive, because lime absorbs carbon dioxide to form lime carbonate according to the chemical reaction  $\text{CaO} + \text{CO}_2 = \text{CaCO}_3$ . When the carbon dioxide has been thus removed, the tendency for stains to occur in the marble is greatly diminished.

#### SAND PUMPS.

One good sand pump will supply five gangs. In some of the more improved types of sand pumps a great saving of power is effected by the use of ball bearings.

#### SAWING WITH UNSOUNDNESS.

If the grain of the marble permits, a great saving of material may be effected by sawing parallel with any joints present in the blocks. This can not be done if joints pass diagonally through blocks, and great waste results. Usually joints strike across the grain, and slabs must be sawed parallel with the grain. Thus joints may intersect a majority of the slabs, and sawing with unsoundness may not be permissible. As a rule unsound blocks can be sawed to better advantage into cubic stock than into thin slabs.

#### SAWING CORNER BLOCKS.

The ordinary method of sawing a corner block is to saw diagonally, forming triangular blocks. This method wastes material and necessitates a fourth cut. An improved method is illustrated in figure 20. Saw 1 is disconnected when it reaches *x* and saw 2 is disconnected at *y*. When cuts are made in a direction at right angles, L-shaped blocks are formed as shown.

#### LOADING SAW BEDS.

In early days the saw beds consisted of transverse stationary timbers. Roadways passed between the gangs over which marble blocks were hauled by teams or cables to points near the beds.

The blocks were placed in proper position by means of crowbars. Although greatly improved methods are now used, many of the old-fashioned mills are still in active operation, chiefly because the cost of remodeling is deemed to be too great. With the present high cost of labor, and of maintaining horses, the economy of adhering to the old methods is doubtful. Although the first cost of refitting would undoubtedly be high, there would be a subsequent saving of much time and labor. So many modern mills are now in operation that it would be a simple matter for any mill owner to estimate the loss or gain attendant on remodeling his plant.

#### THE GANG CAR.

The gang car has of late years replaced the stationary bed. The floor of the car constitutes the bed. Blocks may be loaded on the car and the loaded car placed beneath the blades and securely braced.

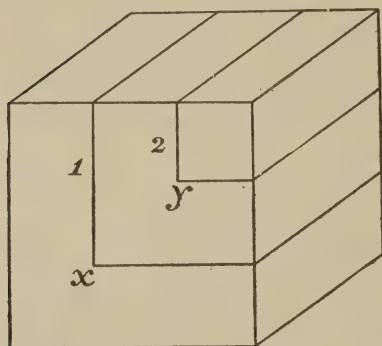


FIGURE 20.—Improved method employed by one marble company for cutting corner blocks.

The loading may be done by means of a derrick, a locomotive crane, or an overhead traveling crane. This method permits the loading of blocks and the removal of sawn material with great facility.

#### THE TRANSFER CAR.

The use of a "transfer car" increases the facility of movement, and is a great timesaver especially where a traveling crane is not used. For small mills a convenient arrangement is to place a row

of gangs on one side of the transfer-car tracks and a platform on the other. On the transfer car are cross tracks for the gang car. These tracks are level with the gang-car tracks which run beneath the saws on the one side and also level with several similar tracks on the platform at the other side. When a block is sawed, the gang car is hauled out by means of a cable and placed on the transfer car, from which it may be transferred to a position on the platform or taken to the shop. Another gang car with a fresh block may then be placed beneath the saws.

The platform is a great convenience when night shifts are employed. Gang cars loaded and ready are placed on the platform. When the cutting of a block is finished during the night it is removed and a new one placed beneath the saws in a short time. If no such conveniences are provided a gang that reaches the bottom of a block during the night must remain idle until morning.

The transfer car is not used at some plants, especially around mills where an overhead crane is employed. The gang must then remain idle during the time the gang car is being unloaded of slabs and reloaded with a new block. Where a transfer-car system is employed, the car loaded with slabs may be shifted to one side and replaced by another car already loaded and waiting.

The wisdom of using both transfer cars and a crane depends on circumstances. With extremely hard marbles, in which the saws sink only 3 or 4 inches during a shift, the time of loading and unloading is proportionally such a small part of the whole time devoted to the sawing operation that any equipment designed to shorten this time may not be justified. On the other hand, with soft marbles in which the saws advance 1 to 2 inches an hour a slow method of removing sawed material and replacing it with rough blocks may result in a considerable proportion of the saws being idle all the time. Mills have been observed where one-fourth to one-third of the gangs were kept permanently in idleness on account of the time required for unloading and reloading. In another mill equipped with both traveling crane and transfer cars, of the 20 gangs, 19 were in active operation at the time the mill was visited. The transfer car is undoubtedly a timesaver, and is not expensive in first cost, operation, or maintenance.

Some mills are so conveniently situated that the gang cars are loaded directly by the quarry derrick.

#### AN EXAMPLE OF A WELL-EQUIPPED PLANT.

The mill shown in figure 18 is an example of a mill suitably arranged and equipped for rapid handling of material. This mill has 43 gangs, and hence it is imperative to have ready facility in exchanging gang cars. The depressed transfer-car track passes down the center of the mill between two rows of saw gangs. When a block has been sawed, a small locomotive crane enters the mill from the yard. It pushes ahead of it an empty transfer car and hauls behind it another transfer car supporting a gang car loaded with a new block. When the empty transfer car reaches the proper position, the car with sawed material is hauled from beneath the saws by means of a cable from the crane and placed on the waiting transfer car. The locomotive crane then moves along until the gang car loaded with a fresh block is in proper position, when it in turn is hauled beneath the saws. The crane then passes on through the mill into the shop, where the sawed material is disposed of.

#### UNLOADING SAW BEDS.

Cubic stock may be removed by derrick or crane by using grab hooks or smooth-faced iron clamps which automatically close upon the block when under tension. Thin stock may be removed in the



same way or by cable slings. The removal of thin slabs singly, especially if no transfer-car system is employed, results in great loss of time and keeps saw gangs idle for too great a part of the time. If a pair of steel cable slings is used the whole load of slabs may be removed at a single operation. Some companies use the slings also for cubic stock.

#### A CONVENIENT DEVICE ON CABLE SLINGS.

The kinking and twisting of steel-cable slings causes annoyance, loss of time, and, occasionally, injury to the operator. A simple device used by one company overcomes this difficulty. A round iron ball weight is attached to the bottom of the sling. This holds the sling under sufficient tension to prevent twisting or kinking. The ball also affords a convenient means of handling the cable. It is less destructive to gloves and less liable to cause injury to hands than the cable itself.

#### THE WIRE SAW.

A useful adjunct to the mill is a yard equipment for cutting with a wire saw. Four to 12 blocks may be placed in line and sawed simultaneously, sand or crushed steel being employed as abrasives. The operation requires little power or attention and gives satisfactory results in uniform material if slight variation in the thickness of the slabs may be allowed. The wire saw will not give as satisfactory service with marble that contains flint balls or otherwise lacks uniform hardness as with pure and uniform material.

#### SHOP OR FINISHING PLANT.

##### LOCATION OF SHOP.

A company which operates both a mill and a shop should logically have the latter so situated that sawed material may be brought to it with the greatest facility. The mill shown in figure 18 illustrates a convenient arrangement, the shop being a continuation of the mill building. A parallel arrangement of mill and shop with a traveling crane between them is convenient, and many companies have their mills and finishing plants arranged in this manner.

##### COPING.

Although most coping is done by hand, a coping machine is employed in a few places. It consists of a small carborundum or carborundum-faced cutting wheel and a smoothly traveling bed on which the slabs are placed. It cuts rapidly and leaves a smooth surface. It may be used for cutting baseboards and tile. It gives good satisfaction if sound stock is employed. One advantage is that no edge rubbing is necessary after cutting with the machine.

## RUBBING.

The two important requisites for a rubbing bed are hardness and uniformity. A good quality of iron has been found most satisfactory. The attempt to make carborundum beds of large size has not as yet been successful, owing to the limited size of carborundum furnaces. Composite carborundum beds could be constructed by joining together a number of segments. However, there would be difficulty in obtaining exactly the same degree of resistance to abrasion in each section. Small carborundum beds are used to some extent for rubbing small pieces.

The common abrasive on rubbing beds is sand. Unless the sand is nearly pure and uniform the use of a simple rotary screen to take out the pebbles is advisable.

The bed surface is kept true by grinding down the high parts with an iron weight. A cubical block of iron resting on the surface of the revolving bed serves this purpose. An improved form of truer consists of cylindrical rotating disks which are adjustable on the frame.

Most rubbing beds are driven from a countershaft and gears above the bed. When driven in this manner the shaft support both above and below the bed keeps the latter in a true horizontal plane. Occasionally, however, beds are geared underneath. It is more difficult to keep the bed running true when driven in this manner, as there is no support above the bed. However, the absence of a shaft above the bed allows great freedom in using a jib crane in handling blocks of marble.

Various methods of rubbing tile are employed to make them true to size and exactly square. Machines for holding them and automatically grinding them true are known to most millmen, and descriptions may be obtained from the various manufacturing companies. One method is to attach 8 to 12 tile together with plaster of Paris and then to rub to size in the same manner that a cubical block is treated.

In order to rub the surfaces of blocks too large to be placed on the rubbing bed, carborundum rubbing heads may be operated in the same way as buffer heads. The carborundum plates may be attached to the head with shellac, melted sulphur, or set screws. A star-shaped head with the water supply entering at the center is a new form that gives good satisfaction.

For curved and irregular surfaces hand rubbing is necessary. A piece of marble with sand and water or a carborundum brick is usually employed.

## GRITTING AND BUFFING.

When a polished surface is desired, the marble is placed beneath a buffer to which is attached a head of felt or other soft texture. Emery powder is used for gritting, and the so-called "putty powder"

for polishing. The polishing powder is composed of chromium oxide, which makes a green powder, or of tin oxide, forming a white powder, together with oxalic acid. Occasionally these powders are mixed.

Different marbles act differently under the buffer. It is only by experience that one can learn the best method of polishing, and the peculiarity of the marble may demand a modification of the polishing machinery. As an illustration, a certain Vermont marble polishes best when nearly dry. The ordinary buffer when nearly dry has a tendency to jump and break thin stock. To overcome this difficulty, "pendulum buffers" have been devised. By means of a crank and pitman large wooden blocks are made to slide back and forth. These blocks are covered with felt pads, by means of which the polishing is done, putty powder almost dry being used.

For hand polishing of curved or irregular surfaces a fine sandstone or hone is employed.

#### CUBIC STOCK.

Unsound blocks cut to better advantage into cubic than into thin stock. On the other hand acute-angled blocks should be cut into thin stock to avoid waste.

For cutting cubic stock to proper dimensions, a perforated steel circular saw, a diamond circular saw, or a single blade in a straight-cut gang frame may be employed. With the perforated steel circular saw sand or steel shot is employed as abrasive. It cuts fairly well but is now replaced in many places by the more rapidly cutting diamond saw.

Circular diamond saws vary in diameter from 20 to 72 inches. The diamond teeth are replaceable. The first cost is high, though with care the cost of maintenance is not excessive. They occupy little space and saw rapidly. An abundance of water is necessary for successful operation, and care must be exercised to avoid overcrowding. A New York company operates a pair of parallel diamond saws which are adjustable for width.

A single saw blade with crushed steel or sand as abrasive is occasionally used. Its operation is too slow to be satisfactory. A blade set with diamond teeth and placed in a straight-cut gang frame may be employed.

#### PLANERS.

Planers are used extensively for cutting moldings and cornices. The more improved forms may be applied to either straight or circular work. Of late years a great deal of the work formerly done with planers is performed with carborundum machines.



## CARBORUNDUM MACHINES.

Carborundum machines are great time savers. The extensive use of carborundum is one of the most remarkable modifications in the equipment of modern marble-finishing shops. Curved work, moldings, cornices, or balusters are all cut successfully with carborundum wheels.

The method of preparing a carborundum wheel is to set it in a lathe, and by means of a steel tool to cut it into the shape of a negative of the pattern desired. The wheel is then placed on the shaft. The marble block travels on the machine bed beneath the wheel which cuts the molding to the desired shape. A copious stream of water plays on the cutting surface. Plate IX<sup>a</sup> shows the manner in which a molding is cut. Round bases may be carved by rotation of the marble mass, and curved forms may be made by a continuous automatic adjustment of the cutting wheel, as shown in Plate X, A. A carborundum wheel in use for making balusters is shown in Plate X, B. The piece of marble is roughed out to the general shape desired. The carborundum wheel and the baluster are rotated in opposite directions. Balusters are manufactured much more rapidly by this method than with a lathe, especially if the marble is hard or tough. One company manufacturing balusters from the Roxbury, Vt., verd antique completes in about one hour a baluster  $3\frac{1}{2}$  feet long and about 6 inches in diameter at its widest part. The time required to turn a similar baluster on a lathe is about three hours.

Machines are now manufactured for fluting large columns. Rather than cut out all the material with a carborundum wheel, one company finds it advantageous to first use diamond saws to make a series of parallel cuts in the deepest part of the fluting. The intervening marble is broken out with a hammer and the operation completed with the carborundum wheel. By thus using diamond saw and hammer to remove the bulk of the material the time required for fluting is just one-half that required if all the cutting is done with the carborundum wheel.

Another company has introduced a simple machine to save time in cutting circular or other irregular work. A horizontal iron bed about 5 feet in circumference is surrounded by a galvanized-iron pan to catch the drip. Projecting from the center of the bed is a vertical shaft to which a carborundum wheel is attached. The marble block is roughed out and the desired outline carefully marked. In order to facilitate movement of the marble block on the bed, flat iron plates 12 by  $1\frac{1}{2}$  inches by  $\frac{1}{4}$  inch are provided with ball bearings inserted in

<sup>a</sup> Pl. IX and Pl. X, A and B, were kindly supplied by the Julius Wegner Machine Works, Astoria, Long Island, N. Y.

holes and projecting slightly above the surface. The arrangement of ball bearings is shown in figure 21. The block is placed on these plates and is thus very easily guided. It is held against the rotating carborundum wheel and cut to the line. The claim is made that one man can accomplish as much with this machine as 10 men cutting by hand.

One weakness of the method is the lumpy or uneven surface that inevitably results from hand manipulation. The company contemplates the introduction of an adjustable block support to be attached to a lathe bed running beneath a carborundum wheel. The bed is designed to travel very slowly while the turning of a handwheel raises or lowers the block. Thus the accuracy of machine work will replace the unavoidable inaccuracy of hand labor. Similarly, the introduction into marble-finishing plants of many new and special forms of carborundum machines may be expected.

#### COLUMN CUTTING.

Various methods are employed for cutting columns of marble. One company cuts small columns by means of a circular steel drum

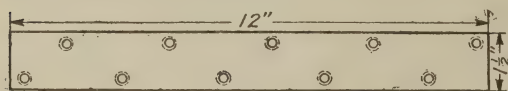


FIGURE 21.—Iron plate with ball bearings used to facilitate movement of marble blocks.

rotating on a vertical axis and fed with sand and water. Other companies employ similar drums having diamond teeth inserted on the

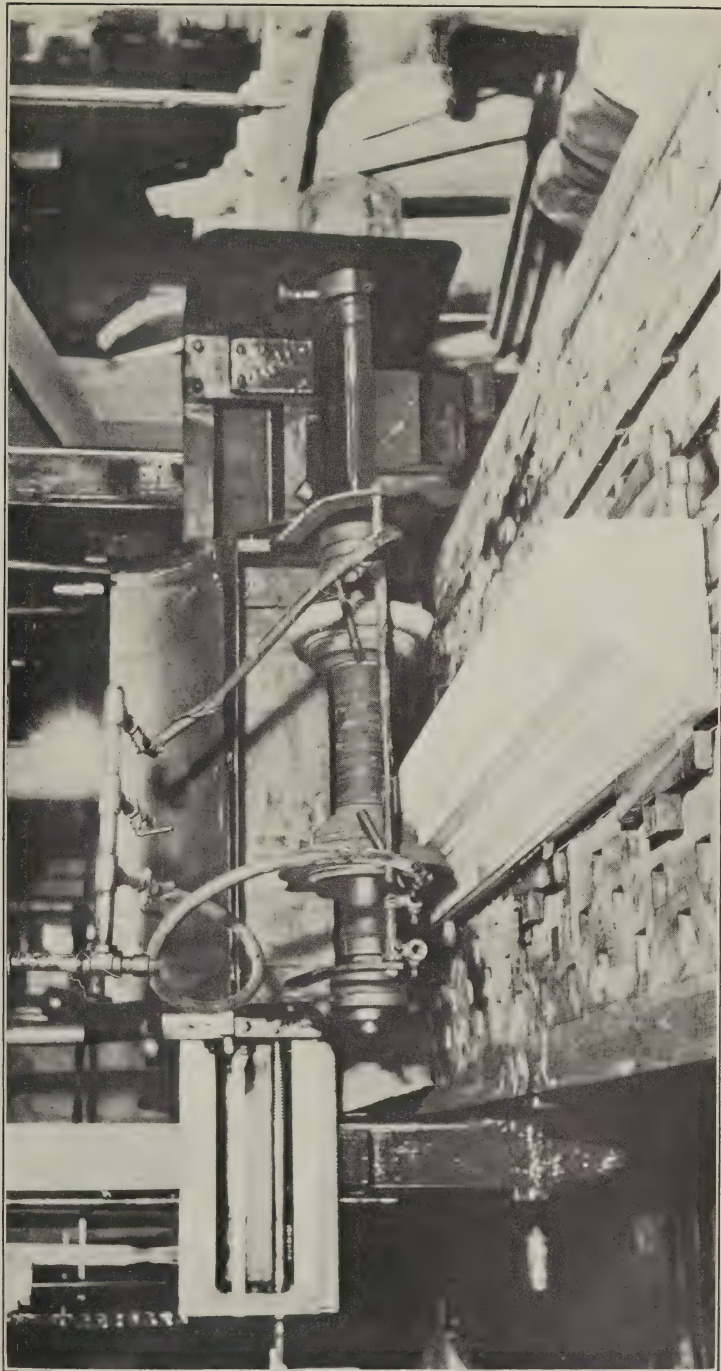
lower margin. The cutting is done by the diamonds and no other abrasive is necessary. A stream of water is supplied during the cutting process.

The largest cutting drum of this type yet observed is that employed by a Colorado company in cutting columns for the Lincoln Memorial, now under construction at Washington, D. C. Each section of a column is 7 feet 5 inches in diameter and 58 inches long. The drum has 80 diamond teeth. A period of 4 to 5 hours is required for cutting each section.

Drum column cutters give good satisfaction for short columns or short sections of columns, as described above. For large monoliths a lathe must be employed. Columns are then roughed out to the approximate form desired before they are placed in the lathe. The cutting tool employed is similar to that used in the ordinary machine lathe for turning metal shafts.

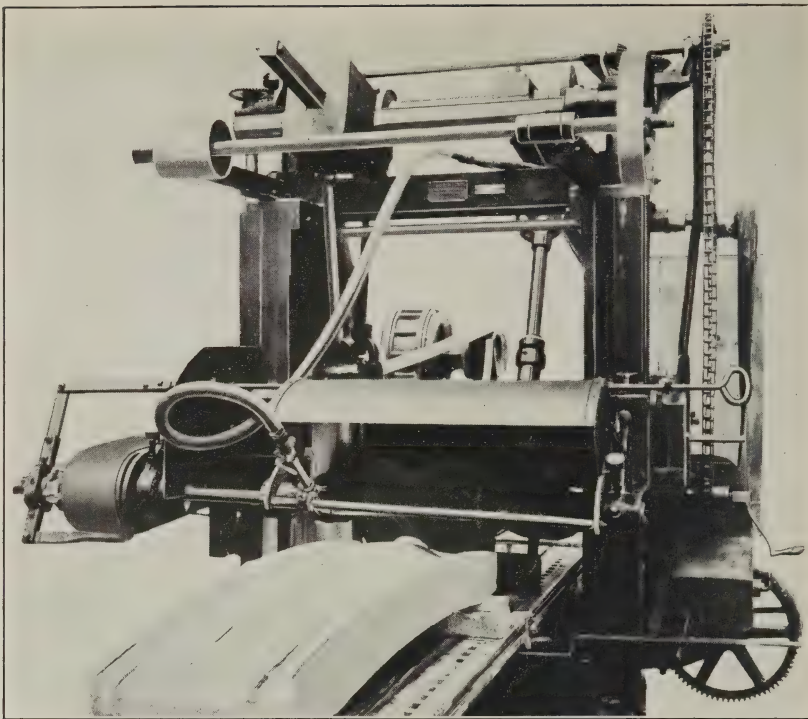
Lathes adapted for fluting as well as turning are now used by some companies.

A lathe may also be employed for polishing plain columns, though the rubbing or polishing of fluted columns is done by hand.

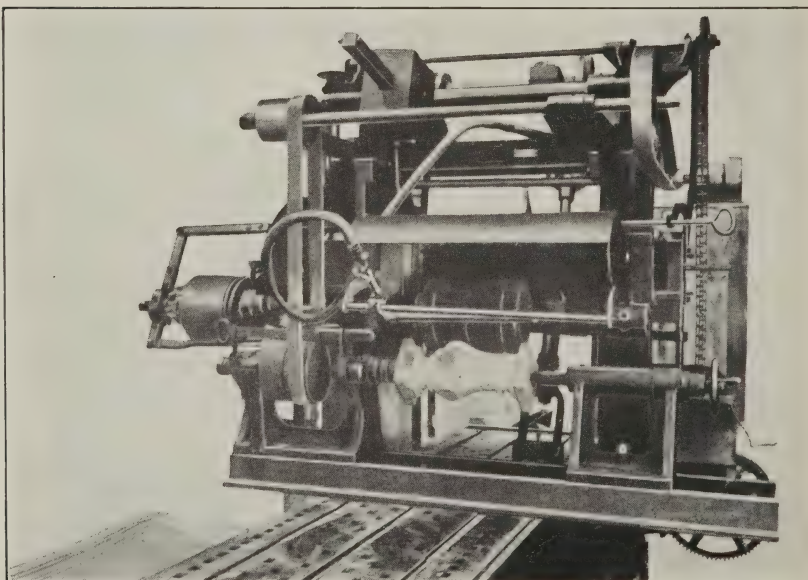


CARBORUNDUM MACHINE CUTTING A MOLDING IN MARBLE.





A. A CARBORUNDUM MACHINE ADAPTED FOR CUTTING CURVED MOLDINGS.



B. A CARBORUNDUM MACHINE ADAPTED FOR CUTTING BALUSTERS.

## CUTTING AND CARVING.

All complicated patterns, or other irregular designs, must be cut by hand. Much of the cornice and molding work formerly done by hand is now performed successfully with planers or carborundum machines. As mentioned previously, carborundum machines are of especial interest to stonecutters, as they are capable of such varied adaptations. For hand carving, small operators with limited capital employ hand tools and hammers. Pneumatic tools are much better, and should be employed wherever possible.

A method of cutting the letters on headstones by means of a sand blast is employed by at least two companies. A shield with an opening the size and shape of the inscription area is placed over the monument. Steel letters are glued on the rock surface in proper position, and a sand blast directed at high pressure against this surface for a few moments cuts down the entire area except that protected by the steel. A little hand trimming is required to correct irregularities due to varying hardness of the stone. Much time is saved by employing this method, especially when many monuments of the same size and shape are manufactured.

That part of a stone shop that is devoted to cutting and carving should be well lighted and heated.

## HANDLING MATERIAL.

An overhead crane is necessary for handling heavy material. Electric cranes are efficient, and are so widely known that description of their operation is superfluous. In many shops where small-sized stock is produced the material is handled with great facility by means of small trucks run by hand.

## CRATING AND SHIPPING.

Experienced men should be employed for crating and loading finished material upon cars ready for transportation. The material must be packed in such a manner that it will not break by moving about in the car, but, on the other hand, it must not be secured too rigidly, or the straining of the cars on sharp curves may cause breakage. For handling heavy material a derrick, locomotive crane, or overhead traveling crane must be employed. For small stock, trucks are commonly used.

## ADEQUATE WATER SUPPLY.

So many operations in the mill and finishing plant require an abundance of water that provision must be made for an adequate supply. Failure of the water supply may make necessary suspension of the operation of part or all of the marble-cutting machinery. If for any reason the supply of water seems liable to diminish and become

inadequate, immediate steps should be taken to obtain a larger supply or to build necessary reservoirs for the regulation and conservation of the supply already at hand.

### **THE PROBLEM OF WASTE.**

#### **THE IMPORTANCE OF WASTE.**

Conservation of national resources demands economy in quarrying. The stone resources of the United States, although great, are by no means inexhaustible, especially the finer grades of marble. Added to the actual value of the material forming the waste heap is the cost of excavation. The cost of stripping, channeling, drilling, and hoisting waste blocks is often nearly as great as the cost of handling an equal amount of good rock. Aside from the value of the rock and the additional cost of quarrying, the waste material encumbers the ground, impedes lateral development, and interferes with yard operations. The failure of some quarry companies to realize a profit is due to the quarrying of an excessive amount of material that remains unutilized.

The problem of waste is twofold. In the first place it has to do with all types of improved equipment and modern methods of excavation which tend to keep the proportion of waste at a minimum; and in the second place it must deal with the various uses to which waste material may be applied. In other words, it is a problem, first, of waste elimination and, second, of the utilization of whatever waste is unavoidable.

#### **ELIMINATION VASTLY MORE IMPORTANT THAN UTILIZATION.**

Waste elimination is much more desirable than waste utilization. A quarryman should by no means countenance methods that result in excessive waste merely because he has found an outlet for his waste material in the form of by-products. As a rule the cash return from by-products is only a fraction of the production cost of the waste material from which they are supplied. As an illustration, it may be assumed that a moderate cost of marble excavation is 25 cents per cubic foot, or \$3 per ton. A fair price for riprap is 50 cents a ton, one-sixth of the cost of excavation. The quarryman seeks a market for riprap, not because the production of riprap is profitable, but for the reason that he prefers to obtain one-sixth of the cost of his waste material rather than to receive nothing at all. By eliminating a ton of waste he saves \$3, whereas by marketing it he saves only 50 cents.

#### **WASTE ELIMINATION.**

##### **UNAVOIDABLE LOSSES.**

The loss of a part of the good stone is unavoidable. Channeling, drilling, scabbling, sawing, and coping are all necessary operations which use up an appreciable share of the stone. In addition to losses



due to the processes of manufacture, more or less stone must be thrown away on account of imperfections.

#### AVOIDABLE LOSSES.

It is, however, the throwing away of masses containing many cubic feet of good stone, or the handling of an excessive amount of inferior material, which constitutes the serious and, for the most part, avoidable losses. Causes of these greater losses and methods of quarrying that will eliminate them are discussed below.

#### CHIEF CAUSES OF WASTE.

The natural imperfections in marble that constitute the source of the greater losses are unsoundness, strain breaks, impurities, and lack of uniformity either in color or texture. On account of the particular method followed in marble quarrying, irregular masses or acute-angled blocks may result and lead to further waste.

#### SYSTEMATIC PROSPECTING.

Unsoundness, texture, and distribution of color or impurities vary from point to point in the same deposit. The prospective quarry operator must not too hastily open a field of operations. Systematic prospecting is a first step toward waste elimination. Outcrops or stripped surfaces should be carefully examined for unsoundness, bad color, flint balls, etc. Naturally, investigation should follow the direction in which any improvement appears in the rock.

The most suitable location as indicated by surface conditions having been chosen, the next step is to ascertain the qualities of the marble at various depths. Drill holes should be projected at points distributed systematically over the area under investigation. Double core-barrel drills are the best for this purpose, as they give information concerning unsoundness in addition to indicating color, uniformity, and supply. Any change in quality with depth is important as showing whether development should best extend laterally or vertically.

#### DIRECTION OF QUARRY WALLS.

Before operations are started the outcrop or stripped surface should be mapped carefully to show the direction of strike and dip, and the directions of the chief joint systems. Naturally the quarry walls should parallel those rock structures that are most pronounced. If the beds are tilted and if inferior beds alternate with those of good quality, it may seem advisable to make the quarry walls parallel the strike and dip. If the rock is of uniform quality but intersected by prominent joint systems, the quarry walls should be parallel and at

right angles to the chief joints, or possibly the contiguous walls should parallel the two chief systems of joints if these should meet at oblique angles. The quarryman must use keen judgment and give careful study to this question. Careful mathematical calculations may be necessary before he can determine definitely which plan will give him the minimum of waste.

The author knows of instances where a mistake in the original plan of quarrying has led to an excessive proportion of waste. In a certain quarry the marble beds strike N.  $35^{\circ}$  W., and dip  $30^{\circ}$ . The sides of the excavation are N.  $60^{\circ}$  W. and N.  $30^{\circ}$  E., respectively. Hence, one wall of the excavation makes an angle of  $25^{\circ}$  with the strike. As a consequence the beds pitch into one corner of the quarry, as shown in Plate VII, *C*. The nature of the rock requires that it be split on the bed, and as a consequence "oblique" blocks result and the waste is excessive.

Of course when such a mistake has been made in the original plan of quarrying, it is possible to change the plan and quarry parallel with the chief rock structures. By such a change, however, corners are left and the original floor space greatly reduced.

#### TUNNELING TO AVOID UPPER INFERIOR BEDS.

The depth of inferior rock due to surface alteration is an important consideration. Although the actual value of the untouched material may be negligible, the cost of handling great quantities of waste material adds greatly to the expense of quarrying. The removal of such material may, under certain conditions, be avoided by employing tunnel methods.

#### WASTE IN TUNNELING.

In projecting a tunnel, head room is obtained by blasting out a space 6 to 7 feet in height. If the tunnel is driven in good marble a large quantity of good material is thus destroyed. If practicable, the quarryman should choose an inferior bed in which to drive the preliminary opening. If a tunnel is to be projected with a view to avoiding handling surface-stained material, it should be driven in the band immediately above the good rock, provided a sound roof of sufficient thickness remains.

The blasting required in tunnel work demands care to avoid shattering the good marble. When tunneling is done on the bed, a band of mica schist or even an open bedding plane at the floor of the preliminary tunnel may serve as a cushion and prevent the shock from affecting the good marble beneath. If no such cushion exists, as a safety precaution, only light charges are used, and holes are fired singly as the simultaneous discharge of a series of shots creates too

severe a shock. The making of a horizontal floor cut with a channeling machine prevents the effects of blasting from reaching the good marble, but the difficulties and inconveniences of the method make it of doubtful value.

#### WASTE DUE TO UNSOUNDNESS.

Channeling regardless of unsoundness probably accounts for the loss of a greater quantity of good marble than any other single cause. Unsoundness is the most prolific source of waste, and the one that is receiving least attention in the majority of American marble quarries. Too great emphasis, therefore, can not be placed on this phase of the waste problem.

Waste results whenever joints pass through blocks, and the waste becomes excessive when they pass in diagonal directions. A reduction to a minimum of this form of waste involves first a modification of channeling and drilling directions in order that they may conform with the directions of the chief joint systems, and second a variation in the spacing of cuts to make them coincide with joints and thus eliminate the joints from the blocks. Attention is directed to figure 10 (p. 63) which illustrates the manner in which waste may be eliminated by a judicious arrangement of channel cuts.

#### WASTE DUE TO LACK OF UNIFORMITY.

If there is a variation in the texture or color or the marble, care should be taken to quarry in such a manner as will tend to produce material that may be closely classified. Thus, channel cuts and drill holes should follow as closely as possible the boundaries between different grades of material. If a block of stone contains 75 per cent of inferior and 25 per cent of good material it will probably be thrown in the waste heap, along with blocks that are 100 per cent inferior. If, however, it is so quarried that even part of the 25 per cent of good material is retained on a good block, a saving has been effected. This feature is discussed on page 60. By making all the cross breaks along lines that mark separations between material of different grades a close classification may be made.

#### WASTE DUE TO IRREGULAR BLOCKS.

Imperfect quarry methods, such as making too few drill holes or wedging so forcibly as to produce artificial strain breaks, may result in the production of irregular blocks. Channeling or otherwise separating blocks along lines making acute angles with open seams or beds may result in the production of irregular or angular fragments, most of which are thrown in the waste heap. When irregular blocks are placed on the saw bed the removal of thick surface slabs



and of irregular or angular ends leaves a relatively small proportion of good stone.

#### THE VARIOUS REGULAR FORMS OF QUARRIED BLOCKS.

Usually marble is quarried in regular blocks bounded by three pairs of parallel faces. Such regular blocks may be in various forms.

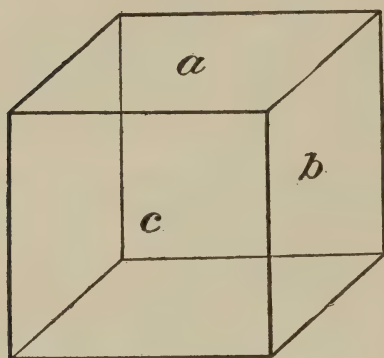


FIGURE 22.—Right-angled block.

The relative waste involved in each form, and the conditions that justify or condemn excavation in such a way as to produce each form are of supreme importance.

Of regular blocks bounded by three pairs of opposite and parallel faces, four different forms are quarried. These are represented in figures 22, 23, 24, and 25. The block represented by figure 22 is bounded by three pairs of parallel faces, all adjacent faces meeting at right angles. Hence, this may be

termed a "right-angled" block. In figures 23, 24, and 25 a right-angled block is shown in dotted lines, in order to indicate the relationship existing between it and the other forms described.

Figure 23 represents a block that results when the quarry floor is level, when one channel cut crosses the line of strike at right angles, and when the cross break or second channel cut is made parallel with a steep dip. The angles between faces *a* and *b*, and between faces *c* and *b*, are right angles, whereas between faces *a* and *c* the angle is greater than  $90^\circ$ . The face *c* and its parallel and opposite face are inclined at an oblique angle with respect to one other pair of faces, and in consequence the block is termed a "monoclinic" block.

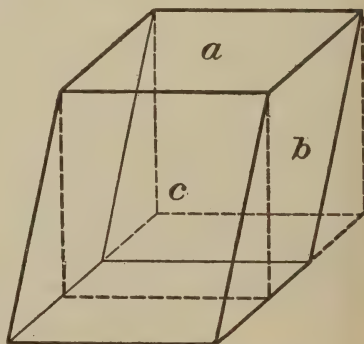


FIGURE 23.—Monoclinic block.

Figure 24 represents the form of block which results when the quarry floor *a* is level, the channel cut forming the face *b* is vertical, and the cross break or channel cut forming the face *c* passes obliquely through the block with respect to both *a* and *b*. Between the faces *a* and *b* the angle is right, whereas the angles between *a* and *c* and

between  $b$  and  $c$  are both oblique. The line marking the emergence of the plane  $c$  on the level quarry floor is not perpendicular to the channel cut that forms the face  $b$ , as with the block shown in figure 23, but passes obliquely across the surface. Hence, this block may be termed "oblique."

Figure 25 shows a block in which no right angles exist between the bounding faces. Such a form results when the quarry floor is level and when the channel cuts are neither at right angles to each other nor vertical, being inclined at steep angles. As there are no right angles and as the three pairs of faces are inclined at oblique angles to each other, such a block may be termed "triclinic."

It may be noted that the four forms described represent the varying stages of obliquity that are possible in regular forms bounded by three pairs of parallel faces. Of the angles formed by the meeting of the three faces  $a$ ,  $b$ , and  $c$  in figure 22, there are three right angles; in figure 23, two right angles and one oblique angle; in figure 24, one right angle and two oblique angles; and in figure 25, all three are oblique.

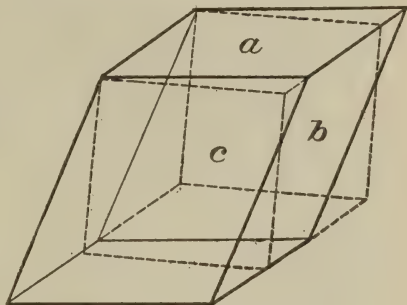


FIGURE 24.—Oblique block.

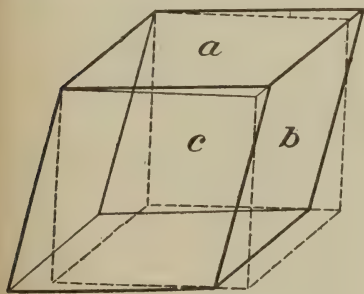


FIGURE 25.—Triclinic block.

#### CIRCUMSTANCES GOVERNING THE PRODUCTION OF THE VARIOUS FORMS.

For convenience in describing the manner in which blocks are formed, the three cuts by which blocks are separated may be termed the "floor cut," the "strike cut," and the "dip cut." The floor cut or "bottom break," as it is sometimes called, is the break that separates the block from the quarry floor, as shown at  $a$ , figure 26. The strike cut is parallel with the strike of the rock as shown at  $b$ , and the dip cut follows a line at right angles to the strike cut, as at  $c$ .

Right-angled blocks are obtained when the quarry floor is level and when the channel cuts and cross breaks are vertical and at right angles. They may also be produced if the quarry floor slants so as to parallel the dip, if the channel cuts and cross breaks are made in such a manner as to form right angles with each other and with the

floor, as shown in figure 26. In some quarries the dip cuts are made with a channeler and the cross breaks by drilling and wedging. If the rift is slight it may be desirable to make channel cuts in both directions. Inclined strike cuts may be made on a slanting floor with Sullivan "Z" or Ingersoll electric air machines. They operate on a level track with the channeling machinery inclined in such a manner as to make the strike cuts perpendicular to the dip as shown in Plate XI.<sup>a</sup>

If beds are inclined at an angle and the quarry floor is maintained on a level it may seem desirable to quarry acute-angled blocks. Inclined marble beds may have a pronounced rift, or a color distribution, or both, in a direction parallel with the bedding. Under such conditions, ease of splitting or a proper grading of material may

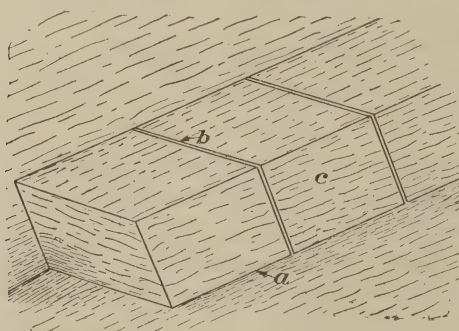


FIGURE 26.—Cuts by which blocks are separated from the solid mass. *a*, Floor cut; *b*, strike cut; *c*, dip cut.

require the strike cuts to be inclined to parallel the dip as shown in Plate XII, *A*. As a result, with a level quarry floor, and one set of channel cuts perpendicular to the strike, "monoclinic" blocks are produced. Such blocks are also formed if the quarry floor is inclined to parallel the beds and the strike cut is channeled vertically.

As mentioned previously, a serious mistake is occasionally made by failing to make the quarry walls conform in direction with the strike of the rock. Under such conditions, inclined open beds will pitch into one corner of the quarry, as shown in Plate VII, *C* (p. 86). When a level floor is maintained, a separation parallel with the beds will pass obliquely with respect to the channel cuts, resulting in what have been termed "oblique" blocks, shown in figure 24. Oblique blocks may also result from a modification of the regular vertical and right-angled direction of channel cuts in order to parallel a system of joints.

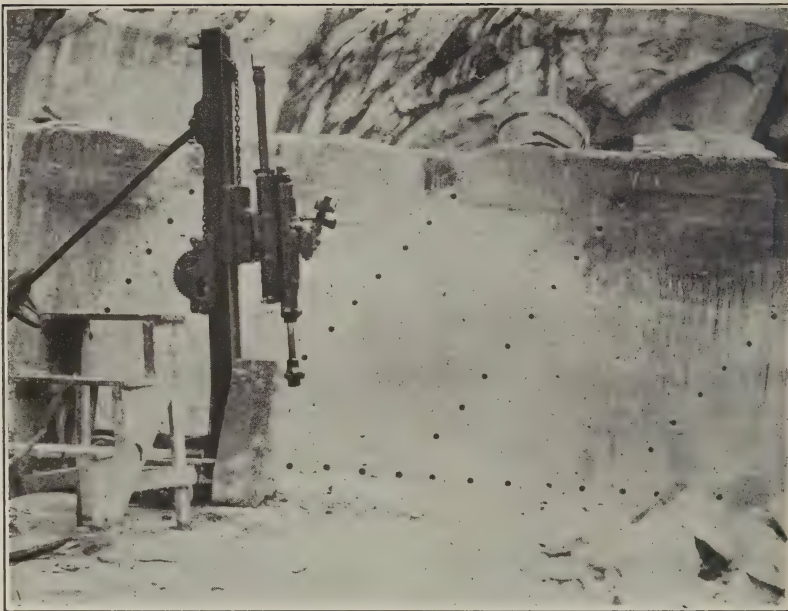
"Triclinic" blocks are produced only under peculiar conditions. In one quarry previously mentioned the marble is quarried on a level floor, the strike cut, which runs N. 5° W., is inclined 60° from the horizontal to parallel the dip, and the second channel cut is made in the direction N. 82° E., and is inclined 61° from the horizontal to parallel a pronounced system of joints. Thus the three pairs of faces

<sup>a</sup> Photograph supplied by the Sullivan Machinery Co.





A METHOD OF CHANNELING RIGHT-ANGLED BLOCKS ON AN INCLINED FLOOR. QUARRY AT AVONDALE, CHESTER COUNTY, PA.



A. METHOD OF QUARRYING MONOCLINIC BLOCKS.



B. VERTICAL DRILL HOLES PARTLY CLOSED BY ROCK EXPANSION.

of the resulting block meet each other at oblique angles to produce a "triclinic" form, as shown in figure 25.

#### WASTE DUE TO ACUTE-ANGLED BLOCKS.

As by far the greater proportion of all structural material is right-angled in form, right-angled blocks are the most desirable and give a minimum of waste material.

When a monoclinic block is cut into cubic stock, angular corners are cut off and thrown into the waste heap. The excessive waste resulting from such a process is indicated in figure 27. Quarrying on an inclined floor, thus producing right-angled blocks, would greatly diminish the waste.

That a greater proportion of waste must result from oblique blocks than from those of monoclinic form is shown in figure 28. If the oblique form results from a mistake in the original layout of the quarry, and if much cubic stock is desired, it is probable that a

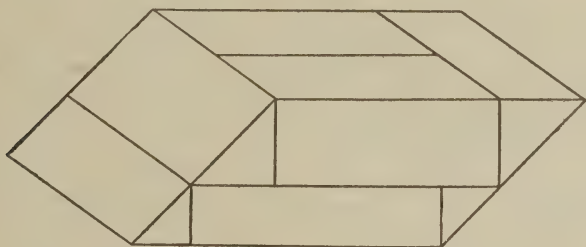


FIGURE 27.—Diagram showing waste resulting from cutting monoclinic blocks into cubic stock.

change in the direction of channeling to conform with the direction of strike would be justified.

No diagram is necessary to indicate that the triclinic block is even less economical than the oblique.

The angle of obliquity has a direct bearing on the economy of quarrying. The acute angles may approach  $90^\circ$  or they may be  $60^\circ$ ,  $45^\circ$ , or even  $30^\circ$ . It is at once evident that the more acute the angle becomes the greater is the proportion of waste. The increase in waste due to increase in obliquity is especially pronounced when cubic stock is produced.

#### JUSTIFICATION FOR ACUTE-ANGLED BLOCKS.

It has been shown that with each increase in the number of oblique angles in a block of sound and uniform marble, provided the acute angles are of approximately the same size, there is a corresponding increase in the proportion of waste. The question arises, Is the production of acute-angled blocks ever justified?



The nature of the product has an important bearing on this question. When the quarried material is cut into thin stock exclusively, slabs of uniform size may be produced from monoclinic blocks with little waste, as shown in figure 29. Except for the beveled edges that must be removed in coping, the waste is not abnormal. However, it must be remembered that in sawing monoclinic blocks longer saws are required than for blocks of the same size in rectangular form. Also, if long blades are used, the cuts are liable to be irregular.

Under certain conditions acute-angled blocks are the most economical that can be quarried, especially if the material to be quarried is unsound or nonuniform. It is illogical to state that as acute-angled blocks do not as a rule cut to advantage their production should always be avoided. Usually economy demands that blocks be separated parallel with the prevailing rock structures, and these structures may not be in right-angled arrangement. They may be so arranged that by quarrying in conformity with them, monoclinic,

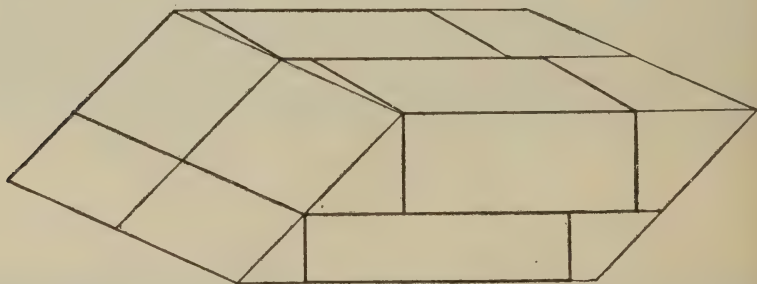


FIGURE 28.—Diagram showing waste resulting from cutting an oblique block into cubic stock.

oblique, or even triclinic forms must result. Rock structures are fixed and unavoidable features, and many of them are undesirable in nature, distribution, and direction, so that the quarryman must modify his methods to make the best of them. Although in quarrying sound marble right-angled forms are the most economical, in quarrying unsound or nonuniform material conformity with structure is of greater consequence than right angles.

A concrete example will illustrate this point. In one part of a certain quarry, color bands and joints make oblique angles with each other and with the level quarry floor. Quarrying parallel with these structures results in the formation of triclinic blocks. Inclining the floor to make a right angle with either the color bands or the joints would result in the formation of oblique rather than triclinic blocks with a consequent small saving of material. However, the conditions described prevail over a small part only of the quarry, and as a consequence a change of floor attitude would not be justified.

If in this instance the quarryman had channeled vertically in two directions at right angles, and thus produced right-angled blocks, the undesirable color bands would have passed through the blocks diagonally in one direction, and the joints would have passed diagonally in another direction. Such blocks would probably be worthless. In other words the right-angled block would give the maximum of waste and the triclinic block the minimum. Thus under certain conditions the quarrying of acute-angled blocks represents the highest type of efficiency.

#### GENERAL RULES GOVERNING SHAPE OF BLOCKS.

Rules for the quarryman's guidance may be summarized as follows:

(1) Effort should be made to produce right-angled blocks, unless there is a valid reason for doing otherwise.

(2) Quarrying on a level floor and splitting diagonally to form monoclinic blocks may be justified where much thin stock is produced. If much cubic stock is desired, the quarryman should consider carefully the advisability of channeling on an inclined floor in order to produce right-angled blocks.

(3) A direction of channeling that results in inclined beds separated by open bedding seams pitching into the corner of a quarry should by all means be avoided. The same is true of inclined beds that are not separated by open seams but have a decided rift or color distribution parallel with the bedding.

(4) As regards unsound or nonuniform material, although an effort should be made to avoid oblique angles, conformity of cuts with structure is, as a rule, more economical than right-angled cuts.

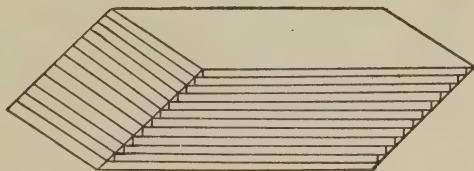


FIGURE 29.—Diagram showing small amount of waste resulting from cutting monoclinic blocks into thin stock.

#### IMPURITIES THAT CAUSE WASTE.

The more common impurities in marble are: Silica, dolomite, pyrite, or marcasite, and mica.

##### SILICA.

The presence of flint or some other form of silica in marble beds is a source of waste in many places. Flint balls or lenses of silicious matter that are seemingly conformable with the bedding are probably original constituents of the marble deposit. As mentioned in the section on marble imperfections, such occurrences are probably characteristic of all parts of the beds in which they appear. The

presence of flint balls in a marble bed should discourage any attempt at extensive excavation with a view to uncovering other parts of the same bed in the hope that flint may not be present. It is wiser to assume that such a bed is undesirable and to conduct quarry operations in such a manner as to handle as little of it as possible. If silica is a secondary filling in cracks and cavities, it tends to follow the joint system, and consequently if the rules given for economical quarrying in unsound marble are followed the silica may be largely eliminated with the unsoundness.

#### DOLOMITE.

Bands or lenses of dolomite are present in some marble deposits. Although dolomite is just as desirable a form of marble as calcite, a mixture of the two is undesirable. Dolomite is harder and less soluble than calcite in ordinary atmospheric reagents. On the other hand, owing to the tendency of dolomite to occur in granular form, it frequently dissolves more rapidly than calcite. On exposed surfaces differential weathering is likely to take place, and in time a pitted or otherwise nonuniform surface will result. Quarrying should be conducted in such a manner as to avoid those beds that show a mixture of calcite and dolomite unless the mixture is intimate.

#### PYRITE AND MARCASITE.

The effect of pyrite and marcasite on the quality of marble may be pronounced or it may be scarcely noticeable. Those forms of either impurity that have a tendency to cause stains by weathering should be avoided in quarrying marble for exterior work.

#### MICA.

The presence of mica in marble is probably the result of the effects of metamorphism on interbedded clay. As clay bands are formed from stream deposits, they are near-shore phenomena. The marble prospector should have a thorough knowledge of the geologic history of his deposit. It is desirable that he should know the position of the shore line when the original beds that formed the marble were deposited. Other factors being equal, it is desirable to locate quarries at some distance from old shore lines in order to avoid as much as possible of the interbedded mica.

#### WASTE DUE TO BAD COLOR.

A great saving may be effected by close grading of nonuniform material. Slight changes of color from white to gray or from white to bluish white may seriously detract from the market value of the stone. The reasons for such changes are obscure. They are probably due to the presence of chemical impurities in small amount.



If the impurities are original components of the marble and due to the peculiar conditions of limestone deposition, it follows, as in the case of flint, that the distribution of color will tend to be fairly constant in any one bed, and the greater variations will be found in passing from one bed to another.

#### WASTE DUE TO STRAIN BREAKS.

A condition of strain within the marble mass has in certain places caused so great a proportion of waste that the workings have been abandoned. The rock is under a severe compressive stress usually in one direction only. The process of quarrying relieves the stress at certain points, and the consequent expansion may cause fracturing. Furthermore the expansion of one mass that is still in rigid connection with the main mass still under compression may cause irregular or oblique fractures to form between the two masses.

In order to avoid the excessive waste due to this cause steps should be taken to bring about relief by uniform expansion of as large a mass as possible at one time. To this end a line of closely spaced deep drill holes should be projected along each side of the quarry parallel with the direction of compression, and a similar line of holes should be projected across the quarry at right angles to the first line. The rock will expand and close the holes in the latter line, and the strain will thereby gain relief. For a more complete discussion of the problem of strain breaks the reader is referred to pages 123-145.

#### WASTE UTILIZATION.

##### BY-PRODUCTS.

Although the proportion of waste may be kept at a minimum by the adoption of economical quarry methods and efficient machinery there is always more or less unavoidable waste. The second phase of the problem of rock waste has to do with the various means of utilizing the waste material. Many manufacturers in various lines of industry have found that the manufacture and sale of by-products from otherwise waste materials have placed their industries on a profitable basis. The tremendous heaps of waste material found near many marble quarries testify to the need of greater development along the line of waste utilization as well as waste avoidance.

The difficulties in the way of such development are of various kinds. The need of a market for rock products hinders all activity in some regions. Local conditions may be such that the demand is small, and freight rates may be excessive. Lack of a market in other places may be due to the fact that although certain rock products are useful, the uses are unknown, and the demand is small until a campaign of public education has created an interest in them.

The nonutilization of waste may be due to lack of equipment. Available capital may not be sufficient to build or purchase the necessary tracks, kilns, crushers, or pulverizers for the manufacture or transportation of the various rock products.

#### RIPRAP.

One important use for waste marble is in riprap for shore protection in rivers and harbors, for filling in piers, or for constructing railroad embankments. The utilization of marble blocks for such purposes depends chiefly on freight rates. The chief item in production is the breaking of massive blocks into fragments of convenient size. This may be done by pop shots for the larger and hand hammers for the smaller blocks.

#### THE BALL BREAKER.

The cost of breaking may be greatly reduced by adopting a method introduced by an Alabama marble company. A breaker consisting of a steel ball weighing about 2 tons is employed. The bale is set in a cavity to avoid breakage. The ball is hoisted to a considerable height with the derrick, then tripped and allowed to fall on the marble block. The impact is sufficient to break large blocks into fragments of convenient size for handling.

An iron sphere weighing about one ton was first employed. It was not strong enough to withstand the severe treatment, and was later replaced by a steel ball of double the weight.

The spherical form first used was unsatisfactory, because the ball would roll and the bale was frequently at the bottom and difficult to reach. A pear-shaped form with the bale at the small end as shown in Plate VI, *C*, is now used. This form tends to keep the bale always in an available position.

The ball can be handled quickly and is less dangerous and less costly than blasting, as the expense of maintenance is low.

#### LIME.

Most marble waste makes excellent lime and a number of quarry companies have constructed batteries of kilns for making lime from the waste. The present and the probable future demand and the price must of course be sufficient to justify the expenditure of the necessary capital for such an enterprise.

#### FLUX.

If smelting works are situated at a convenient distance so that freight rates are not excessive scrap marble may be sold for fluxing purposes.

## IMPROVEMENT OF SOILS.

It is doubtful whether either quarrymen or farmers realize how extensive are the areas of agricultural lands that need liming, or how numerous and diverse are the crops that may be benefited by the application of ground limestone or lime.<sup>a</sup> It is a popular error to assume that soils in limestone or marble areas do not need lime. Many soils that have been formed from limestone, are greatly in need of lime. It is beyond the scope of this bulletin to outline the geographic areas in which lime is needed. Such information may be obtained from the various soil survey reports issued by the Department of Agriculture or by State bureaus carrying on experimental work of a like nature. These reports indicate that large areas of farming lands situated in marble regions are giving mediocre returns, although piles of material that would greatly increase the productiveness of the soil lie unused in the near vicinity.

Government reports state that a judicious liming of the soil would increase by a varying percentage the agricultural productiveness of nearly every State. Consequently, those quarrymen who have vast heaps of waste marble in convenient localities for distribution should attempt a campaign of publicity along such lines as will open this market for their waste products and develop an industry mutually beneficial to both quarrymen and farmers.

Marble waste is being used for soil improvement in a few places, but not nearly as extensively as it might be used. Near quarries at Cockeysville, Md., the upper marble beds are decomposed to a fine powder which is sold as fertilizer without grinding or pulverizing. The slush from marble mills is an excellent material for this purpose, and requires no grinding.

Marble for agricultural purposes should be finely pulverized. The general view is that the finer the material the better. It may be, however, that the expense of producing the finer grades may make it more economical to supply a larger amount of coarser material. It may be stated that at least 80 per cent of the ground rock should pass through a sieve having 100 meshes to the linear inch.

Lime derived from burning limestone or marble in kilns is more effective in the soil than the pulverized rock. Also transportation costs less, as marble loses greatly in weight when burned, although no calcium is lost. There is the same amount of calcium in 2,000 pounds of quicklime as in 3,550 pounds of marble. In other words, 2 tons of pulverized limestone have approximately the same effect on the soil as 1 ton of quicklime. Thus it may be seen that by converting limestone to lime half the freight expense could be eliminated and a more efficient product placed in the market. With the develop-

<sup>a</sup> See Wheeler, H. J., The liming of soils: U. S. Dept. Agr., Farmers' Bull. 77, 1905, 23 pp; the maintenance of fertility; liming the land: Ohio Exp. Sta., Bull. 279, July, 1914, 22 pp.



ment of a wide and active demand for lime fertilizer it may pay to burn the waste marble into lime rather than to pulverize it. Quarrymen who now burn lime from their waste marble should endeavor to develop a market for all waste lime as fertilizer.

#### TERRAZZO.

For terrazzo flooring the crushed rock fragments should be somewhat cubical in shape and there should be a minimum of material in the form of flat flakes or slivers. Before installing a crusher for producing terrazzo, experiments should be made to determine the type of crusher that will give the maximum proportion of material of the desired form. Fine-grained white marble is in greatest demand for terrazzo, though there is a limited demand for black or other dark colors for making borders or patterns.

#### CEMENT.

Although marble waste may under favorable conditions be used for the manufacture of cement, its use for this purpose is not usually practicable. The supply is inadequate for the manufacture of cement on a profitable scale. Only where cement plants are situated near marble quarries, and can use the marble waste to supplement their limestone supply, can such a means of utilization be profitably employed.

#### ROAD MATERIAL.

Crushed marble alone makes a good though not a very durable road. It pulverizes easily and, unless oil or asphaltic material is used, blows away as dust. When mixed with clay it is more durable. It can be used to advantage if more permanent materials are not available.

#### RUBBLE.

A limited amount of marble is used for foundation stone and retaining walls. Considerable labor is required to reduce the masses to convenient size and shape. The demand is in most places merely local and therefore limited.

#### MISCELLANEOUS.

Inferior waste material may be used to advantage for local improvement of quarry property. Hollows may be filled and the yard brought to a general level, new roadways may be constructed, or dams built to form water reservoirs.

Certain quarries that use buckwheat coal for fuel, and are situated near large cities, sell their cinders to be used in place of gravel for concrete construction. This is a good illustration of the profitable use of material that many quarrymen would consider to be absolute waste.

**STRAIN BREAKS IN QUARRIES.****LOSSES DUE TO STRAIN.**

In many quarries in rocks of both sedimentary and igneous origin, at certain stages in the process of rock removal, expansion of the rock will take place, or irregular fractures will suddenly form, occasionally accompanied by subterranean noises. Such minor earth fracturing has been studied by scientists in an effort to obtain new light on earthquake problems. The economic aspects of the phenomena have been almost disregarded, although to quarrymen they are of prime importance. In many quarries a modification of methods may become necessary owing to the interference of strain with the successful operation of certain quarry tools. Of still greater importance is the fact that in a number of marble quarries the formation of the so-called "strain breaks" results in the waste of much valuable material. In recent years it has become necessary to abandon at least three marble quarries in widely separated localities because the marble which otherwise would have been valuable was rendered useless by excessive strain breaks. Such abandonment has occasioned the loss of thousands of dollars, and the excessive waste from the same cause in many other quarries now being worked has resulted in the loss of many thousands more.

**STRAIN CAUSES WASTE, NOT IMPAIRED QUALITY.**

As reference is subsequently made to particular quarries, one point must be emphasized in order that marble dealers or others may have no misconceptions concerning the effect of "strain" on finished marble. So far as is known, the so-called "condition of strain" in a quarry in no wise detracts from the value of the finished product either in strength or durability. Whatever damage strain breaks cause to the rock is accomplished before the blocks leave the quarry floor and manifests itself in fracturing and rending masses into irregular forms which can not be worked to advantage. The strain causes waste and not impaired quality.

**EXTENT AND IMPORTANCE OF STRAIN.**

The occurrence of strain breaks is more general than was at first supposed. In view of the heavy losses sustained by several quarry companies, an inquiry into the manifestations and probable origin of strain breaks has been undertaken, and as a result of such investigations new quarry methods are suggested tending toward a reduction in the proportion of waste from this cause. Various instances of strain phenomena recorded in literature are given in brief abstract in chronological order, and to these are added examples from the writer's personal observations.

**EXAMPLES OF STRAIN PHENOMENA RECORDED IN LITERATURE.**

One of the earliest references to the phenomenon is given in "The Miners Dictionary," by William Hooson, 1747, in which several instances are given of subterranean noises which have greatly alarmed workmen. No attempt at explanation is given. The author states, "Miners say that the knocking is some Being that Inhabits in the Concaves and Hollows of the Earth; and that it is thus kind to some men of suitable tempers, and directs them to the Ore by such its knocking."

In 1854 Johnson <sup>a</sup> described strain in the Portland sandstone quarries near Middletown, Conn. If a uniform channel was cut across the beds in an east-west direction, when the opening reached a point near the bottom of a bed, pressure acting in a north-south direction would suddenly crush into fragments the remaining part. An expansion in the rock mass of three-fourths of an inch is mentioned, but no reference is made to the length of the mass that thus expanded. When a bed was channeled north and south no such compression could be observed. The author states that the rocks are "not perfectly at ease in their ancient bed," but attempts no explanation.

In 1871 Niles <sup>b</sup> described movements and expansions of gneiss as noted in a quarry near Monson, Mass. The strike of the planes of foliation is given as N. 10° E. and the dip as N. 80° W. at an angle of 80°. Cross joints are rare. The rock appears to be under compression in the direction of strike. It was found that fracturing could be prevented by channeling the rock across the strike. A mass of rock 354 feet in length expanded 1½ inches when broken loose. The formation of fractures was accompanied by loud reports, and the projection of dust and rock fragments into the air was noted.

These phenomena are later described by the same author <sup>c</sup> in greater detail. Additional evidence is presented that the rock discussed is clearly under compression and, when freed, expands.

Three years later Niles <sup>d</sup> described a condition of strain in various localities and deduced certain conclusions from these occurrences. The records are briefly summarized in the following paragraphs.

Niles states that in the sandstone quarries of Berea, Ohio, there is evidence of a strong horizontal compressive stress in a north-south direction but none east and west. The process of quarrying requires

<sup>a</sup> Johnson, John, Notice of some spontaneous movements occasionally observed in the sandstone strata in one of the quarries at Portland, Conn.: Proc. Am. Assn. Adv. Sci., vol. 8, 1854, p. 283.

<sup>b</sup> Niles, W. H., Some interesting phenomena observed in quarrying: Proc. Bos. Soc. Nat. Hist., vol. 14, 1871, p. 80.

<sup>c</sup> Niles, W. H., On some expansions, movements, and fractures of rocks observed at Monson, Mass.: Proc. Am. Assn. Adv. Sci., vol. 22, 1873, pt. 2, p. 156.

<sup>d</sup> Niles, W. H., The geological agency of lateral pressure exhibited by certain movements of rocks: Proc. Bos. Soc. Nat. Hist., vol. 18, 1875-76, p. 272.



channels to be cut at intervals. In quarries with a northerly and southerly working face the channels are cut east and west. When a bed has been channeled nearly through, the stone at the bottom of the channel cut is broken and crushed by the lateral pressure, and the width of the channel is perceptibly decreased by expansion of the adjacent rock. As such crushing renders a portion of the stone worthless for structural purposes, it was found advisable to cut the trenches in short sections in order that the crushing should be confined to limited masses. The expansion and fracturing has been noted at all seasons of the year and hence is seemingly independent of temperature changes. The belief has been general that the pressure was produced by the weight of the adjacent overlying rock. The author points out, however, that the lateral compression observed could not result from vertical pressure on adjacent parts of the beds.

In the Lemont (Ill.) limestone quarries it was noted that in certain places where potholes crossed the boundary between adjacent beds, as a result of the slipping of one bed on the other the upper parts of the holes were offset with respect to the lower parts. An arching of the rock in an anticlinal fold with east-and-west axis was also observed. This again indicated compression in a north-and-south direction. These movements do not perceptibly vary with temperature changes.

In a granite gneiss quarry at Waterford, Conn., where drill holes were made very near each other, the intervening parts of rock were often crushed, and, by expansion of the rock, the drill was so pinched that it could not be operated. The compressive force was in a northeast-and-southwest direction.

Evidences of compression were also noted in quarries at Groton, Conn.

The number and distribution of these occurrences lead the author to conclude that they are due, not to local or external forces nor to peculiarities of composition or metamorphism of the rock mass itself, but to "the continued action of the same geological power which has been the chief agency in the elevation of continents and mountain systems." He likens these fractures and movements to earthquake shocks on a small scale. The tendency toward a north-and-south direction of compressive stress points to a great continental stress in this direction. The gentle undulations that may be noted in the alternate elevation and subsidence of the Atlantic coast from Greenland to Florida are in his opinion the evidence of slight foldings resulting from this compression.

Gilbert <sup>a</sup> discovered several postglacial anticlinal ridges in the horizontal limestones of Jefferson County, N. Y., and in the slates near

<sup>a</sup> Gilbert, G. K.. An account of some new geologic wrinkles: *Am. Jour. Sci.*, 3d ser., vol. 32, 1886, p. 324.

Dunkirk, in western New York. He attributed the expansion that caused such arching to the warming up of the surface rocks as they recovered from the cold of the glacial period.

Strahan <sup>a</sup> describes what he terms "explosive slickensides" in certain lead mines of Derbyshire, England. The walls of the lead veins are rubbed and polished as though moved while in violent contact. When struck or scratched with a pick the wall in many places will break or explode with violence, throwing blocks to a considerable distance. Undercutting of the vein wall has the same effect. His conclusion is as follows: "The explanation which perhaps best satisfies the requirements of the problem appears to be that the spars are in a state of molecular strain, resembling that of the Rupert's drop, or of tempered glass, and that this condition of strain is the result of the earth movements which produced the slickensides."

Hughes <sup>b</sup> refers to similar phenomena in a limestone quarry at Dent Head, and a tunnel at Ribble Head, in Yorkshire, England. He attributes the fracturing to the fact that the limestone rested on shale. When an excavation was made leaving only a thin layer of limestone over the shale, the weight of the overlying limestone on either side was sufficient to cause the shale to flow, after the manner of a viscous fluid, toward the region of reduced pressure. This caused the rock to arch up in the bottom of the quarry. When the brittle rock was thus arched it would break and fly when further excavation was attempted.

Gresley <sup>c</sup> describes explosive fracturing of coal in the mines of South Staffordshire, England. The explosions are termed "bumps," and are said to be of frequent occurrence during the process of excavation, causing many tons of coal to fall down at the working face. Various subterranean rumblings have also been noted. He attributes such phenomena to "the upsetting by the excavation of the equilibrium of the strains or pressure holding everything fast and firm together—the removal of the support, thereby causing the rocks to get relief and to fly off or apparently to explode." He terms such explosions as "miniature earthquakes."

Cramer <sup>d</sup> describes an interesting anticlinal uplift in the bed of the Lower Fox River near Appleton, Wis., which took place immediately beneath a long stone mill, resulting in an arching of the floor and roof with consequent displacement of machinery and cracking of the walls. Two miles distant, at Kaukauna, Wis., the rock between two parallel joints 30 feet apart became arched as a result of compressive stresses. The rock in a zone about 2 feet wide on each side

<sup>a</sup> Strahan, Aubrey, On explosive slickensides. *Geol. Mag.*, vol. 4, Dec. 3, 1887, p. 400.

<sup>b</sup> Hughes, T. McK., Bursting rock surfaces, *Geol. Mag.*, vol. 4, Dec. 3, 1887, p. 511.

<sup>c</sup> Gresley, W. S., Re "Explosive slickensides:" *Geol. Mag.*, vol. 4, Dec. 3, 1887, p. 522.

<sup>d</sup> Cramer, Frank, On recent rock flexure: *Amer. Jour. Sci.*, ser. 3, vol. 39, 1890, p. 220.

of the joints was splintered by the pressure. No explanation is presented, though the author refers to Gilbert's suggestion that such phenomena may be due to expansion caused by a rise in temperature after the disappearance of ice from glacial regions.

Reade <sup>a</sup> discusses various references to strain. He cites many instances of recent changes of level in the surface of the earth and concludes that from "the prevalence of these vertical movements in recent geologic times it is obvious that the subsidence of a low arch of elevation must tend to put the surface rocks into lateral compression." This state of compression may be influenced by the presence of joints and faults. The fact that conditions of lateral compression are recorded as unusual raises a strong presumption that tangential thrust arising from a shrinkage of the earth's nucleus is not the agent. He concludes therefore that the phenomena are due to the changes of level mentioned above.

A more complete examination of the anticlinal arch at Appleton, Wis., was obtained when the water was removed from part of it and a further description was given by Cramer <sup>b</sup> in 1891. He records several other instances of strain breaks in quarries of that vicinity. The direction of fracturing is highly variable, and seems to be determined rather by the means of relief than by the preponderance of pressure. He concludes that the rock is under compression "in all directions." A set of anticlinal ridges parallel with the strike of the Corniferous limestone beds at Lime Rock in Genesee County, N. Y., are, however, attributed to rock collapse produced by solution and removal of salt and gypsum from the underlying beds.

Gilbert <sup>c</sup> describes anticlinal ridges 6 to 8 feet high in the Devonian shale of Ripley township, in western New York. He notes similar anticlinals in the Devonian shale of northwestern Ohio and in the Trenton limestone in northern New York. He reiterates his former contention that all are to be attributed to expansion due to post-glacial rise in temperature.

Matthew <sup>d</sup> describes numerous thrust faults in the slates near St. John, New Brunswick, the most probable cause of which he gives as lateral thrust from the southeast.

Chalmers <sup>e</sup> describes numerous thrust faults in the slates of southeastern Quebec. They all occur near mountains or resisting masses

<sup>a</sup> Reade, T. M., The cause of active compressive stress in rocks and recent rock flexures: *Amer. Jour. Sci.*, ser. 3, vol. 41, 1891, p. 409.

<sup>b</sup> Cramer, Frank, On the rock fracture at the combined locks mill, Appleton, Wis.: *Am. Jour. Sci.*, ser. 3, vol. 41, 1891, pp. 432-434.

<sup>c</sup> Gilbert, G. K., Postglacial anticlinal ridges near Ripley and Caledonia, N. Y.: *Am. Geol.*, vol. 8, 1891, pp. 230-231.

<sup>d</sup> Matthew, G. F., Movements of the earth's crust at St. John, New Brunswick, in postglacial times: *New Brunswick Nat. Hist. Soc. Bull.* 12, 1894, pp. 34-42.

<sup>e</sup> Chalmers, R., Report on the surface geology and auriferous deposits of southeastern Quebec: *Geol. Survey of Can. Ann. Rep.*, vol. 10, 1898, pt. J, pp. 9-12.



of rock and are attributed to a pushing of the beds against such masses, to a sinking of the resisting masses by cooling and contraction, or to both.

Campbell <sup>a</sup> has described a fold in sandstone which he attributes to processes of weathering. The increase in volume by weathering causes a forcing apart of joints and cleavage fissures. The process is assisted by thermal expansion and contraction by freezing of water in the joints and by the force of growing roots. The cumulative effect of such forces in many joints produces rupture and arching at a point of weakness.

Beard <sup>b</sup> intimates that gas escapes into coal mines as a result of earth movement, and in this connection shows that many mine explosions are contemporaneous with earthquake shocks.

Woodworth <sup>c</sup> states that thrust faults of recent date occurred near Troy, N. Y. He notes several brick and stone structures resting on the rock, which are fractured and faulted, showing that faulting has occurred since the structures were built. Similar faults were noted in Rensselaer, Defreestville, and Pumpkin Hollow. The whole series is attributed to a mountain-building thrust similar to and in the same direction as that which formed the mountains of the region. Reference is made also to postglacial faults in Quebec, Massachusetts, and New Hampshire.

Dale <sup>d</sup> describes a state of compression in granite quarries of Waldo, Hancock, Kennebec, and Lincoln Counties, Me. He also cites the observations of Gilbert at Lithonia, Ga., and reproduces a photograph of an anticlinal arch formed by compressive stresses in the granite of this region. The same author <sup>e</sup> describes the arching of Vermont granite under compression. Under a later date he describes <sup>f</sup> in detail the compressive stresses in various Vermont granites, and gives the direction of compression of each. At Bethel the direction was east-west, at Barre, chiefly north-south, at Woodbury, northeast-southwest, at Groton, in all directions, at Ryegate, east-west, at Dummerston, N. 10° E. to S. 10° W. The effect of the strain is to close channels and crush cores between drill holes.

Van Horn <sup>g</sup> refers to a series of anticlines in the Chagrin shales of Cleveland, Ohio, which he attributes to the increase in volume that results when iron sulphide contained in the shales alters to iron sulphate and alum-like compounds.

<sup>a</sup> Campbell, D. F., Rock folds due to weathering: Jour. Geol., vol. 14, 1906, pp. 718-721.

<sup>b</sup> Beard, J. T., Colliery explosions and their causes: Eng. and Min. Jour., vol. 83, 1907, pp. 1051-55.

<sup>c</sup> Woodworth, J. B., Postglacial faults of eastern New York: New York State Museum Bull. 107, 1907, pp. 5-28.

<sup>d</sup> Dale, T. N., The granites of Maine: U. S. Geol. Survey Bull. 313, 1907, pp. 34, 42, 121, 142.

<sup>e</sup> Dale, T. N., Chief commercial granites of Massachusetts: U. S. Geol. Survey Bull. 354, 1908, p. 25, B.

<sup>f</sup> Dale, T. N., The granites of Vermont: U. S. Geol. Survey Bull. 404, 1910, pp. 17-18.

<sup>g</sup> Van Horn, F. R., Local anticlines in the Chagrin shales at Cleveland, Ohio: Geol. Soc. Am. Bull., vol. 21, 1910, pp. 771-773.

Lawson<sup>a</sup> describes a series of postglacial thrust faults in Archean slates near Banning, Ont. He attributes these and similar thrust faults to volume change in the surface rocks brought about by chemical or temperature changes.

#### EXAMPLES OF STRAIN PHENOMENA OBSERVED IN FIELD WORK.

Other examples of strain phenomena noted by the writer during the field season of 1914 are described below.

In a quarry near Knoxville, Tenn., the strike of the marble beds is N. 55° E., and the dip 35° to 40° SE. Slip joints appear parallel or nearly parallel with the bedding. In certain parts of the quarry zones of parallel joints run N. 40° to 50° W. The rock is subjected to a severe compressive stress approximately parallel with the strike. Great difficulty is experienced in separating the blocks without irregular fractures. Much valuable stone is lost by fracturing into angular pieces during the process of quarrying. A distinct though small earthquake shock is reported by a quarryman to have occurred a few years ago within half a mile of this quarry. This was probably only another manifestation of the condition of strain within the rock. In this instance it exceeded the elastic limit, and sudden fracture and displacement occurred.

A blue-marble quarry situated about a mile and a half north of Florence, Vt., has had some trouble with strain breaks though no details were procurable. Other quarries near the same locality have suffered severely from the same cause. One quarry was abandoned because the strain breaks destroyed so much rock that excavation could not be conducted with profit. The rock is above the average in soundness, masses 35 feet in length having been quarried for columns.

A quarry near Clarendon, Vt., had to be abandoned on account of excessive strain breaks. When the author visited the quarry the pit was full of water. The structure as described by Dale<sup>b</sup> is as follows: The beds strike N. 10° W. and dip 42° W. At least one bed is crossed by a slip cleavage. Joints strike N. 35° W., and dip 45° to 55° W. The rock has a distinct rift parallel with the bedding. When an opening of considerable depth was made near the eastern end of the excavation a violent fracture occurred which shook houses in the vicinity and frightened the workmen so that they could not be induced to reenter the pit for several days. The fractures took place toward the west and thus ran downward on the rift, destroying the rock to such an extent that the pit was abandoned.

<sup>a</sup> Lawson, A. C., On some postglacial faults near Banning, Ont.; Seism. Soc. Amer. Bull., vol. 1, December, 1911, p. 159.

<sup>b</sup> Dale, T. N., The commercial marbles of western Vermont: U. S. Geol. Survey Bull. 521, 1912, p. 112.

A quarry not far distant was affected by similar, though less extensive, strain breaks. In this quarry the beds strike N.  $55^{\circ}$  to  $60^{\circ}$  E., and dip  $25^{\circ}$  N.  $33^{\circ}$  W. They are crossed by cleavage dipping  $25^{\circ}$  E. The direction of strain was not ascertained.

The granite at Vinal Haven, Me., lies in sheets 2 to 5 feet thick. Vertical joints are rare. In quarrying out a floor it has been found that if all material is removed from the upper surface of a sheet 3 feet thick and 50 feet long, the sheet will arch at least three-fourths of an inch in the center. When one end is set free by channeling it will settle back into its original position.

In a quarry near Woodbury Center, Vt., a condition of strain resulted in a minor accident. The projection of a row of horizontal drill holes formed a plane of weakness, and owing to internal strain a mass of rock broke loose and struck a driller's hand with sufficient force to cause a flesh wound.

At Stonington, Me., in certain granite quarries the channel-bar method can not be employed to obtain a heading, as when holes are drilled close together the strain is sufficient to crush the parts of rock between the holes and to jam the drills. To avoid this difficulty a 2-foot channel is made by drilling and blasting. It is stated that when a mass of rock 50 feet long is set free at one end it will stretch three-fourths of an inch.

At Lithonia, Ga., the granite forms anticlinal arches by compression. One of these arches observed by the author was about 30 feet across and was raised so as to leave an open space beneath about 14 inches high.

#### GENERAL ANALYSIS OF STRAIN BREAKS RECORDED.

An analysis of all the examples noted and the explanations given by various authors points to the conclusion that by far the greater majority of all serious strain breaks in quarries are in reality minor earthquakes and must therefore be attributed to the same causes as earthquakes. The case recorded above of a strain break in a Vermont marble quarry of such intensity that it shook structures in the immediate vicinity is a striking example of the identity of strain breaks with earthquakes.

It seems probable that strain breaks in general are not to be attributed to local causes but to great and far-reaching forces. Most of the explanations given above are in accordance with this hypothesis. Explanations opposed to the hypothesis are as follows:

Both Gilbert and Lawson give as a probable explanation a rise in temperature following the removal of glacial ice. Although this is a possible cause, and may be contributory, its acceptance as the chief cause is discouraged by the fact that in regions in Georgia and



Tennessee that are beyond the limits of glaciation are strain breaks seemingly similar to those in New England.

Hughes's theory of rock flowage in a plastic layer beneath a limestone bed is of local application only, as in most stone quarries no such plastic layer exists.

Cramer's theory of displacement by collapse due to subterranean solution is also of local application only.

The hypotheses of Campbell, of Lawson, and of Van Horn that buckling is due to expansion as a result of the chemical or physical changes which accompany weathering may apply to a few localized superficial strains, but not to a condition of strain at depth.

In addition to the hypotheses mentioned above it seems probable that temperature changes may cause local expansion of rock, as for example the arching of granite, which has been noted at Lithonia, Ga. Merrill <sup>a</sup> refers to the results of tests made at the Watertown Arsenal which indicate that rocks do not completely recover after expansion, but assume a permanent "set." Repeated expansions and contractions by diurnal or seasonal changes in temperature may therefore cause a gradual expansion of surface layers with consequent arching. Changes of temperature are slight at a short distance beneath the surface, and consequently it is unlikely that a state of compression deep within quarries can be attributed to this cause.

All the other explanations offered in the preceding pages have a more or less direct bearing on the theory that the majority of strain breaks are due to the great forces of nature that build mountains and continents, and produce earthquakes.

Dale <sup>b</sup> states that the cause of the strain observed in certain eastern Tennessee marble quarries is the same as that which produced the Appalachian folds, and which operated both parallel and at right angles to the axes of the folds, the former only being operative at present.

#### RELATION OF EARTHQUAKES TO STRAIN BREAKS.

Milne <sup>c</sup> divides earthquakes into two groups, the macroseismic or great disturbances, and the microseismic or minor shocks. The latter are described as "settlements and adjustments along lines of their (the greater earthquake's) primary fractures," and are confined to areas of a few miles' radius. He estimates the number of the latter as at least 30,000 a year for the entire earth.

Hobbs <sup>d</sup> has pointed out that constant adjustments and changes of level are taking place in the earth's crust, as evidenced by post-

<sup>a</sup> Merrill, G. P., *Stones for building and decoration*, 1903, p. 478.

<sup>b</sup> Dale, T. N., *The marbles of eastern Tennessee*, unpublished manuscript.

<sup>c</sup> Milne, J., *Seismological observations and earth physics: Geog. Jour.*, vol. 21, 1903, p. 1.

<sup>d</sup> Hobbs, W. H., *Earthquakes*, 1907, pp. 211-223.

glacial faults and known variations in altitude of certain geographic areas, with respect to sea level. Such gradual changes are termed "bradysisms." He refers to the common occurrence of subterranean rumblings, "brontidi," which are not accompanied by sensible earthquake shocks. These are attributed to slow adjustments of the earth's crust, such adjustments taking place as frequently repeated slight displacements rather than as gradual movements.

Chamberlain <sup>a</sup> claims that the earth acts like an elastic, rigid, crystalline body in which strains accumulate until they reach an intensive stage, and then yield either to slow movements of great magnitude, as illustrated by various changes in level, or by sudden, swift, non-continuous movements of less magnitude, as earthquakes.

Gilbert <sup>b</sup> states that earthquakes are of two types, as follows: (1) The tectonic, those arising from subterranean mountain building forces, and (2) the volcanic, those resulting from the movements of lavas. The former are the more important, especially in the United States. The tectonic process is briefly described by him as follows:

In the formation of mountains and other great features of the earth the rock masses are forced into new shapes. They are pulled, pushed, twisted, and bent, so that strata, for example, which were originally flat, become inclined and curved. If the changes are sufficiently slow, the component particles of the rock readjust themselves gradually; but if the changes are comparatively rapid, the rocks are broken. Before fracture occurs there is elastic yielding or "strain;" that is, the rock is compressed or stretched or bent somewhat like a spring; and when its strength is at last overcome the dis severed parts recoil. This recoil is instantaneous, violent, and powerful, and is in the nature of a jar.

As a result of observations of earthquakes in Panama, MacDonald <sup>c</sup> concludes that they are manifestations of relief from strain. The noises resembling ice fractures on a cold morning appeared to be "due to the formation of small shears or strain-relieving cracks in the rocks."

#### ORIGIN OF THE STRESSES.

Deformation of the earth's crust is constantly taking place. Acting under the force of gravity the earth's shell suffers collapse and buckling. According to the isostatic hypothesis it is assumed that the continents with their plateaus and mountains are composed of and deeply underlaid by lighter matter than the sea bottoms and therefore stand higher because they float higher. A variation in load due to erosion and deposition causes an undercurrent or flow of rock material from the area loaded toward the area denuded. Opposed to the isostatic theory is that of high rigidity within the earth's crust,

<sup>a</sup> Chamberlain, T. C., Diastrophism and the formative processes: Jour. Geol., vol. 21, November-December, 1913, p. 681.

<sup>b</sup> Gilbert, G. K., The cause and nature of earthquakes: Min. Sci. Press, vol. 92, April 28, 1906, p. 272.

<sup>c</sup> MacDonald, D. F., Some earthquake phenomena noted in Panama: Science, new series, vol. 41, May, 1915, p. 783.

a rigidity sufficient to sustain mountains and continents in higher position than sea bottoms merely by lateral thrust.

According to the former hypothesis the flow of rock from one point to another is resisted by rock rigidity, and a condition of internal strain results. Dutton <sup>a</sup> states that one of the probable causes of many earthquakes is the tendency of the earth to recover its isostatic equilibrium when, by denudation, transportation, and deposition of vast masses of material, this condition of equilibrium is destroyed. The theory of high rigidity, on the other hand, assumes that thrust rather than flow causes internal strain. Whatever may be the correct theory the fact remains that changes of level are of constant occurrence, and strains varying in intensity and direction are the result of such adjustments.

### ACTION OF ROCKS UNDER STRESSES.

In order to describe the behavior of rocks under stress some principles of earth physics must be emphasized. Hoskins <sup>b</sup> states that "a body is strained when the relative positions of its particles undergo any change." If the limit of elasticity is exceeded, the change will become permanent—that is, the body will not assume its original form when the force is removed. This permanent change of form may find expression in fracture or in flowage, depending upon plasticity. If fracture results, earthquakes of greater or lesser intensity are produced.

When plasticity overcomes elasticity flowage will take place rather than fracture. This condition is illustrated by the somewhat intricate folding in many mountainous regions. Chamberlain <sup>c</sup> has shown that deformation having the aspect of plastic deformation may take place by a process of granulation and progressive recrystallization. Such a process undoubtedly does take place in marble, as evidenced by the extreme recrystallization.

The matter of supreme importance in this discussion is, however, the state and action of the rock under stress within the elastic limit. If the limit of elasticity has not been exceeded the body will recover its natural configuration when the stress is removed. That rocks are both compressible and highly elastic has been clearly shown by the numerous examples of automatic recovery from strain referred to in previous pages, and has been proven in the laboratory by the experiments of Adams and Coker.<sup>d</sup> A condition of strain within the elastic limit is that condition which quarrymen call "strain." The

<sup>a</sup> Dutton, C. E., *Earthquakes*, 1904, p. 37.

<sup>b</sup> Hoskins, L. M., *Flow and fracture of rocks as related to structure*: Sixteenth Ann. Rept. U. S. Geol. Survey, pt. 1, 1896, p. 860.

<sup>c</sup> Chamberlain, T. C., *Diastrophism and the formative processes*: Jour. Geol., vol. 21, 26, p. 678.

<sup>d</sup> Adams, F. D., and Coker, E. G., *An investigation into the elastic constants of rocks, more especially with reference to cubic compressibility*: Carnegie Inst., Wash., Pub. 46, 1906, p. 16.



latter term includes also all permanent deformations. For the sake of simplicity the term "strain" is subsequently used in the sense of strain within the elastic limit. This is the condition of rock masses that has led to such disastrous losses in many marble quarries.

The exact relation of elasticity to plasticity of rocks under varying conditions has a direct bearing on the permanence of strain. Rudzki<sup>a</sup> states that the elastic force with which a body resists a deforming force diminishes with time. Hobbs<sup>b</sup> in applying this principle to rock folds concludes that the period of time covered by the evolution of a fold may be sufficiently long to permit the resistance within the mass to fall. If this conclusion is correct, we may expect that after a great lapse of time plasticity will overcome elasticity and equilibrium will be attained by folding without fracture. Folding may thus take place within what is usually termed the zone of fracture if the process is sufficiently gradual. If, however, nature's process hastens the operation beyond a certain "speed limit," fracture must result.

Thus it would appear that after a tremendous lapse of time permanent deformation may take the place of strain, and a state of perfect equilibrium be attained. No strain breaks would occur in quarries situated in such masses. It might be concluded that marble deposits that have not visibly suffered mountain-building movements since Cambrian or other early geologic age would have attained such a state of rest, and would therefore be free from strain.

It is possible that strain is more common in regions where intense folding is of comparatively recent age. The examples of destructive strain in marble so far observed are all to be found in the Appalachian belt, which was folded toward the end of the Paleozoic era. No hard and fast rule can be given, however, because, as already pointed out, ever-varying earth movements may cause internal stresses anywhere and at any time.

Rudzki's principle further states that temperature has a decided influence on plasticity, an increase in temperature resulting in a corresponding increase in plasticity. The effect of overburden is to increase the temperature and therefore to increase the plasticity. Denudation lowers the temperature, decreases the plasticity, and therefore increases the tendency to fracturing rather than to deformation by flowage. In surface rocks, consequently, the relaxation time is great, and a condition of strain may exist ages after the initial force has been exerted if the force tending toward deformation has remained constant.

The foregoing discussion indicates that a strain may be inherited from a former geologic era, possibly accounting for a condition of

<sup>a</sup> Rudzki, M. P., *Physik der Erde*, Leipzig, 1911, pp. 232-233.

<sup>b</sup> Hobbs, W. H., *Mechanics of formation of arcuate mountains*: *Jour. Geol.* vol. 22, 1914, pp. 193-194.

strain in some isolated rock hills to which it seems impossible to some scientists that present forces can transmit their effects. It is conceivable also that rocks now at the surface were formerly deeply buried and subjected to such intense pressure from superincumbent material that they were forced to assume a smaller volume. Processes of denudation gradually brought these masses nearer to the surface, where the residual strain might find expression in shock fracture, or where a means of relief in the form of strain breaks was obtained through quarry operations.

#### THE LIMITATIONS OF STRAIN.

As these forces are so far-reaching and constant, it might be expected that strains would be almost universal. One explanation that they are not is the limited cubic compressibility of rocks. When strain in a granite is relieved, an expansion of three-fourths of an inch over a length of 50 feet is perhaps a fair illustration of the compression possible without rupture. It is evident, therefore, that little lateral movement is required to bring relief. Most rocks are intersected by joint planes, which permit the necessary movement to give relief from pressure. It is significant that quarries having considerable unsoundness have little or no strain. There is, therefore, some consolation in the fact that these two great evils are not likely to occur in the same quarry.

Strain in one direction only has been noted in a number of localities. In some places the great earth stresses may be in one direction only. In others it seems probable that strain is produced because rocks can find no relief in one direction, but are free in all other directions. It is significant that as regards most rocks having a steep dip, the strain is parallel with the strike and no strain is apparent across the strike.<sup>a</sup> A slipping of the beds one upon another may give the necessary relief in the transverse direction, whereas parallel with the strike the absence of joints or planes of weakness may afford no such slipping. In a Tennessee quarry the strain is approximately parallel with the strike and there is evidence of slipping parallel with the beds, which gives the necessary relief in the transverse direction.

#### EFFECT OF QUARRYING ON STRAIN.

In quarries in which strain exists the internal strain-stress relations may be complex. Various forces are acting and reacting, and their total result is a condition approximating equilibrium. The fact that a condition of strain exists within the limit of elasticity indicates that true equilibrium has not been attained. According to Rudzki's principle, the elastic resistance of the rock is gradually

---

<sup>a</sup> Dale, T. N., The marbles of eastern Tennessee, unpublished manuscript.

diminishing and slow permanent deformation is constantly going on. As a result of excavation the approximate equilibrium is thrown out of balance locally. If we could imagine all stresses except the downward force of gravity ceasing simultaneously, we could conceive of the strained rock recovering its natural configuration without rupture. Even then the realignment of particles in a thick bed could scarcely take place rapidly without fracture. In the stone quarry the conditions are more extreme. The strain-stress relations are disturbed in a limited region only.

Before excavation is begun a stress that tends to deform the rock is balanced by the resistance of the rock plus the weight or lateral pressure of other rock masses. When these masses are removed the resistance of the rock alone is not sufficient to withstand the deforming stress, and fracture results. This is clearly illustrated by the examples previously given of the crushing of rock at the bottom of channels in the Portland and Berea sandstone quarries (pp. 124-125). Such fractures are the result of compression.

Strain may manifest itself in other ways. At some stage in the process of cutting a block of marble free the support that holds it in compression may no longer be sufficient and the block expands. As it is still rigidly connected with the main rock mass, which remains under compression, no considerable expansion can take place without fracturing between the two masses. Such fracturing may be in the nature of shearing, with an even break between the two masses and lateral displacement.

Thus it is established by theory, which unfortunately has been confirmed by practice in many places, that in quarries where strain exists the successive separation of small masses from the main quarry rock generally results in the formation of fractures, producing excessive waste. Some new method of quarrying must be found if this waste is to be reduced.

#### BASIC PRINCIPLES LOOKING TO REDUCTION OF DAMAGE FROM STRAINS.

The following basic principles looking to a solution of the problem have been established: (1) The causes of strain are not to be found in mere local conditions such as the weight of a near-by hill or the process of surface weathering. Strains are manifestations of the great geologic forces that build mountains and cause earthquakes. (2) In every observed instance of strain the rock has been under compression. (3) The compression is usually in one definite direction. (4) The only way to remove the strain is to bring about expansion of the rock to its normal volume. (5) If the strain on a small mass of rock is relieved, it expands independently of the main mass, with consequent fracturing; therefore relief from strain should be sought for a large mass at one time.



## THE REMEDY.

The principles outlined indicate that some means should be found to cause uniform expansion of a large mass of rock at one time in such a manner as to reduce to a minimum the occurrence of irregular fractures between the mass set free and the main quarry rock. When the strain has been relieved the mass can be quarried without fear of fractures. The first step to be made in quarrying rock under strain is to determine the direction of compression. In most instances quarrying will already have been done, as otherwise the condition of strain will not have been revealed. It is usually easy to determine the direction of compression by noting the direction of movement of the loosened blocks with reference to the main mass. The lateral displacement of drill holes or other marks will indicate the direction.

In order to bring about the necessary relief an opening should be made at right angles to the direction of compression. Expansion of the rock will manifest itself by a gradual narrowing of this space, and the rock will thereby find relief from strain. The opening should be of sufficient width to allow complete relief by expansion and of sufficient length and depth to free a large mass of rock at one time.

A channeling machine would probably give poor results, as an open cut gives too ready a means of relief. The pressure would tend to close the cut immediately and to jam the bits, and only a small mass of rock would be freed. Also the opening should in most instances be deeper than a channeling machine can cut. Drilling a row of close holes would probably afford the best means of relief. The compressive force would gradually crush the rock remaining between the drill holes, close up the holes, and thus relieve the strain. The presence of these supporting cores would prevent a too rapid closing, and would thus allow uninterrupted drilling.

If the direction of compression is parallel or approximately parallel with the quarry walls the quarryman is fortunate. Rows of deep vertical holes should be drilled along both quarry walls parallel with this direction. The holes should be deep enough to pass entirely through the bed if the latter is horizontal or nearly so, and if open or weak bedding planes occur. In massive or steeply inclined beds the holes should be at least 12 feet, and preferably 15 or 20 feet deep. Convenient sizes are  $1\frac{3}{4}$  or 2 inches in diameter. The width of rock intervening between the drill holes should average not more than 1 inch.

When the rows have been completed, a row of holes of similar depth and spacing should be projected in a line at right angles. This third row may be placed at either end of the quarry or in some intermediate position. The latter is probably more effective especially if the compression is not exactly at right angles to the line of drill holes, as motion in opposite directions in both masses will afford a more ready means of the cores slipping on each other.

The purpose of the two lines of holes parallel with the direction of compression is to give an easy shear line when expansion takes place. When they are drilled before the cross line of holes, no motion occurs to jam the drills. If, however, they are drilled after the cross line the tendency of the rock to offset by shearing would continually interfere with drilling operations.

The transverse line of holes should be projected rapidly. Several drills should be employed in order to complete the line as nearly as possible before any appreciable closing of the holes takes place.

Care should be taken that no excessively wide cores are left in the corners. The holes should be well cleaned out and covered in some way to keep out *débris*. Tightly driven hardwood plugs may tend to prevent the desired closing of the holes. Plugs of cotton waste or similar material may be used.

Along the transverse line a crushing of the cores and a closing of the drill holes will indicate expansion of the rock mass. There will probably be some lateral motion, permitting the cores to slip by each other and project into the open holes.

Along the shear lines expansion of the rock will not appreciably close the holes, but will by lateral motion offset one-half of each hole with respect to the other half. The extent of lateral displacement will diminish with distance from the transverse row of holes.

What may occur at the bottom of a mass of rock that is thus permitted to expand is as yet by no means certain. It is possible that fracturing may take place, and fractures so made will tend to occupy diagonal positions, causing much waste. It is for the purpose of reducing such fractures to a minimum that deep holes are recommended.

If irregular bottom fractures cause excessive waste, or if the expense of making deep holes is considered to be too great, an alternate method is proposed, as follows:

A narrow part of a floor along one side of the quarry parallel with the direction of compression may be first removed. The remainder of the floor may then be quarried as shown in figure 30. A row of horizontal holes along the floor at *a* should be closely spaced. A similar line of vertical holes is then projected at *b*. If the direction of compression is parallel or nearly parallel with the face, it is probable that a channel cut may be substituted for this second line of holes. If now a cross line of holes, *c*, is made at either end of the mass or in some intermediate position, expansion will crush the cores and close the holes. Along the planes *a* and *b* the cores will break and shear. Theoretically, at least, no irregular fractures should occur, as an even plane has been provided on every side. It is impossible to foresee what disturbing elements may arise, but the method holds sufficient promise to justify a trial.

If the direction of compression is diagonal to the quarry walls, it may be necessary to change the directions of channeling and drilling in order that they may more nearly conform with the strain line. This change will lead to a reduction in floor space occasioned by leaving masses in the corners. When the angle between the direction of compression and the direction of a quarry wall is not more than  $15^{\circ}$  or  $20^{\circ}$ , channeling and drilling in the usual directions will probably be satisfactory.

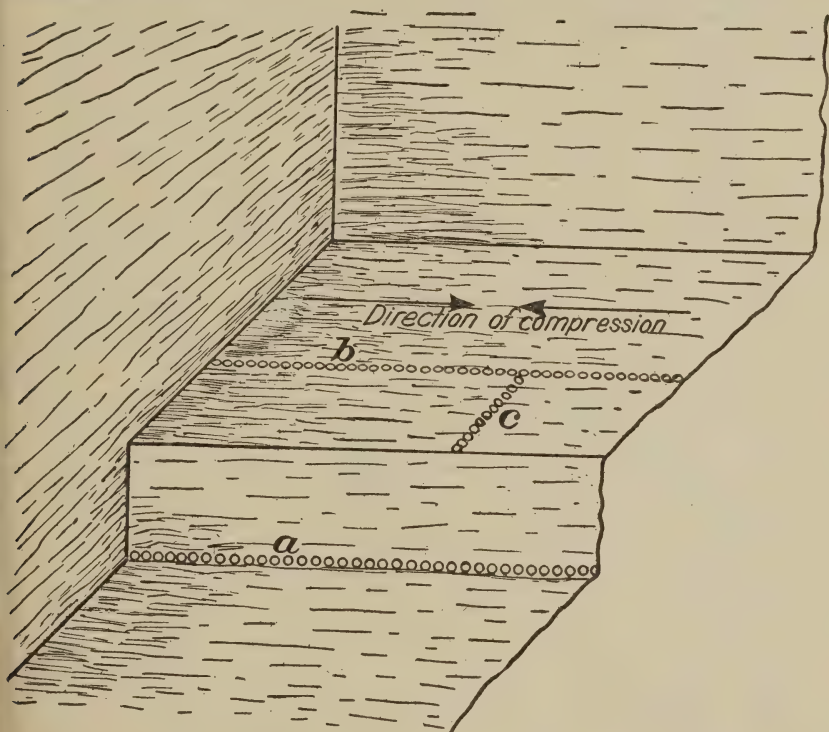


FIGURE 30.—Method of drilling holes to avoid strain breaks.

#### A SUCCESSFUL EXPERIMENT.

A Tennessee marble company, acting on the advice of Mr. T. Nelson Dale, of the United States Geological Survey, who visited the quarry during the summer of 1914, made an effort to relieve strain by drilling holes in accordance with the first plan outlined above. Although defects were apparent in the preliminary trial, the great increase in the proportion of sound stock produced encouraged a continuation of the method. During April, 1915, the writer observed the progress that had been made up to that time and offered some suggestions toward further improving the method. A complete description of the plan of drilling, effect on the rock, and of suggested changes in method follows.



The mass of rock under observation was 69 feet long, 20 feet wide, and 12 feet thick. It occupied a position along one side of the quarry, the remainder of the floor to the northwest having already been removed. Its position is shown at *A* in figure 31.

A line of holes perpendicular to the supposed direction of compression was first drilled. The holes were  $1\frac{3}{4}$  inches in diameter, 12 feet deep, and the average core thickness between drill holes was fifteen sixteenths of an inch. The direction of the line was N.  $40^{\circ}$  W. The line was 20 feet long, extending from the corner of the quarry along the



FIGURE 31.—Plan of part of a marble quarry, showing position of a rock mass studied. *A*, rock mass where observations were made; *a*, position of line of holes driven perpendicular to supposed direction of compression; *b*, position of second line of holes driven to give rock mass freedom to shear on expanding.

northeastern wall and terminating at the excavated part of the floor. Its position is indicated by *a* in figure 31.

A second line of holes of similar depth and spacing was projected along the southeastern wall of the quarry, as shown at *b* in figure 31. The purpose of this line of drill holes was to give the mass of rock freedom to shear when it expanded. On account of the difficulty of placing drills, holes were omitted for a distance of 10 inches from the corner on the northeast wall, and 6 inches from the corner on the southeast wall. As indicated in figure 32, a mass of rock 12 inches wide was left in the corner.

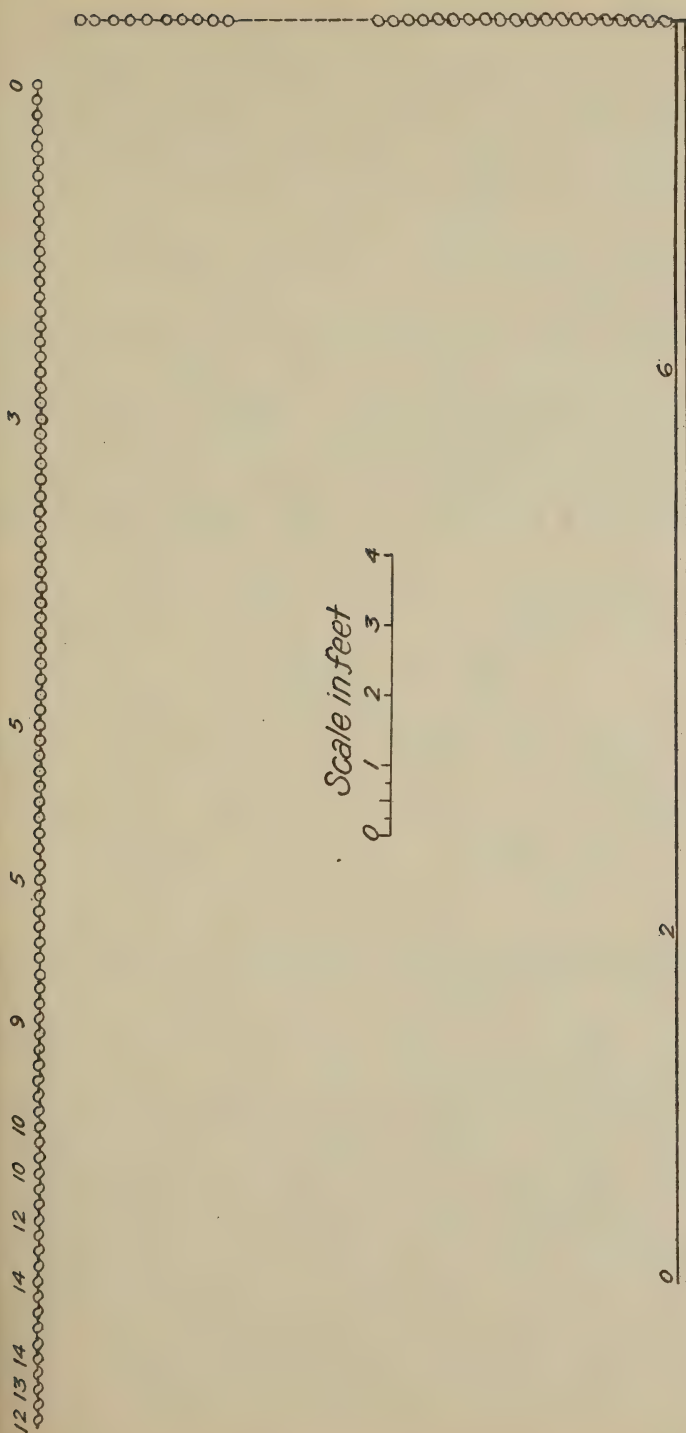


FIGURE 32.—Appearance of drill holes and channel cut after expansion of rock. Size of drill holes and their shape after expansion are drawn accurately to scale. The numbers indicate the degree of closing, in thirty-seconds of an inch, at various points.

Expansion of the marble soon manifested itself by a crushing of the cores and a gradual closing of the drill holes. The closing did not take place uniformly. Near the northwestern end of the line it was greatest, attaining a maximum of seven-sixteenths of an inch, which gradually decreased to zero at the other end of the line. There were 88 drill holes in the 20-foot line. The extent of closing as measured at various points was as follows:

Hole No. ....	1	3	5	10	14	15	17	20	27	36	64	88
Extent of closing in thirty-seconds of an inch. ....	12	13	14	14	12	11	10	10	9	5	3	0

The appearance of the partly closed holes is shown in Plate XII, *B*. The manner in which expansion took place is shown in figure 32. The cores were broken the entire length of the line, though the movement in the last holes was too slight to be recognizable with the naked eye. Likewise on the perpendicular line every core was broken by shearing. However, the mass in the corner was unbroken, as far as could be seen, and it would therefore appear that this mass of rock,

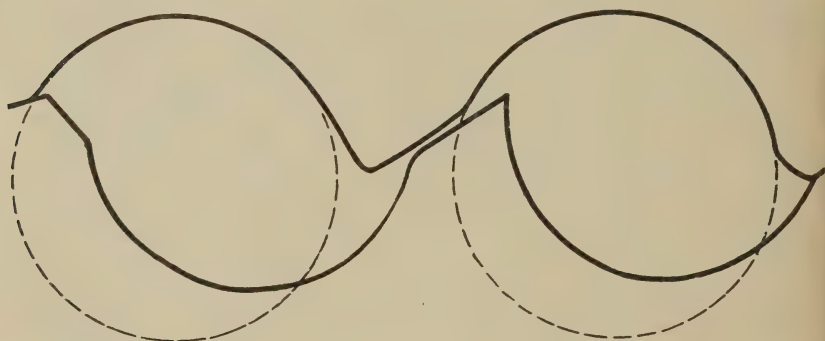


FIGURE 33.—Second and third drill holes from the left end of the line shown in figure 32, illustrating the manner in which the holes were closed by compression. Actual size and shape; dotted lines show original position of drill-hole walls; solid lines show position after rock movement.

which was 12 inches in width, effectively prevented the major part of the mass to expand so as to gain relief.

Figure 33 illustrates the second and third drill holes in the cross line drawn natural size. In the lower part of the sketch the light line represents the original position of the drill hole walls, and the heavy line the position of the same walls subsequent to the expanding movement. It is evident that motion took place in a diagonal direction, which is undoubtedly due, in part at least, to a lateral slipping of the cores in order that they might gain ready relief by projecting into the open holes. However, the fact that slipping took place toward the corner where a mass of solid rock prevented any movement would seem to indicate that the direction of compression was not



exactly perpendicular to the line of drill holes, but inclined at a small angle toward this corner.

At a later time a channel cut was made parallel to the first line of holes and 69 feet distant from it as indicated in figure 32. After the cut had been completed, the fact that complete expansion had not occurred along the first line of drill holes became apparent. That part of the 69-foot mass that had previously been unable to expand eastward on account of the uncut corner began to expand westward and partly closed the channel cut. The shearing accompanying this expansion broke the cores between the drill holes along the entire shear line, and the displacement gradually increased with the distance from the corner.

Measurements were made of the closing of the channel cut. Owing to a shattered condition of the rock, measurements could not be made nearer than 5 feet to the quarry wall. At this point the cut was closed six thirty-seconds of an inch, 13 feet from the wall it was closed two thirty-seconds of an inch, and 18 feet from the wall it was not closed at all.

Thus it is evident that where relief was gained along the original line of drill holes no further relief was necessary, but where in the original line no motion was possible a backward motion took place as soon as a means of relief was provided.

A considerable number of strain breaks are present in this mass of rock. It is reasonable to assume that they were formed in consequence of the nonuniform and imperfect relief obtained, as pointed out above. The part of the mass that had been quarried previous to the time of the author's visit is said to have expanded more uniformly and was remarkably free from strain breaks.

The suggestions following are offered with a view to improving the method of quarrying already outlined.

(1) If it is impossible to complete a line of drill holes in a corner, the corner should be cut across diagonally with a closely spaced row of holes in order that no mass too great to be crushed may remain.

(2) To avoid delay by jamming of the drills, the shear-line holes should be made first, as no appreciable movement of the rock will take place until the cross line of holes is projected.

(3) If drilling costs seem to justify it, holes should be more than 12 feet deep, as the object to be continually kept in view is to gain uniform relief at one time for as large a mass of rock as possible.

### **COST KEEPING.**

#### **BEARING OF COST KEEPING ON SELECTION OF METHODS AND MACHINES.**

In the preceding text many different methods and types of machinery have been mentioned. Certain general advantages or disadvantages have been pointed out, but in many instances no absolute rules could be given regarding the desirability of one over another. In no two quarries are conditions alike. Variations in hardness, structure, texture, and rift, in bedding and attitude, or in unsoundness are extreme. The fact that a day's channeling for one machine varies in different quarries from 20 to 150 square feet is a typical illustration. On this account marble quarrying if conducted efficiently must involve considerable experimental investigation. A quarry operator ought to ascertain which of certain methods or machines are best adapted for his peculiar conditions. The results of past experience may enable satisfactory selection, but often experimentation must be adopted. However, no experiments convey definite information unless careful records of relative costs are kept. Therefore, cost keeping is a necessary part of such experiments. It is the yardstick that is used to measure efficiency.

### **COMMERCIAL RECORDS.**

A certain amount of cost keeping is a recognized necessity for every successful business. Accounts that balance general expense, pay roll, fuel and supplies, maintenance and repairs, interest, and depreciation against receipts from sales, and thus furnish a basis for a statement of loss and gain on any transaction are what may be termed commercial records. Practically all quarrymen keep them in some form, though many such systems are inadequate.

### **TECHNICAL RECORDS.**

Another and even more important phase of accounting is that dealing with detailed cost accounts for all operations in and about the quarry—accounts that are kept to supply a definite and accurate record of the relative efficiency of men, methods, and machines. Such may be termed technical records. They may be in the form of production, time-efficiency, or cost records. Cost records of each operation may be connected with commercial books by means of some suitable intermediary accounts, the nature of which should be determined by a skilled accountant.

### **RELATION OF TECHNICAL RECORDS TO EFFICIENCY.**

Webster defines efficiency as the ratio of useful work, or of the effect produced, to the energy expended in producing it. The process of supplying the required energy costs money, and thus from

the economist's standpoint efficiency is the measure of accomplishment in terms of the cost. Increased efficiency, then, means decreased cost of production. Many books have been written on efficiency, on the relation of efficiency to organization, system, wages, etc. It is not within the scope of this bulletin to enter into all phases of business efficiency. An attempt is made simply to point out the desirability of keeping accurate and systematic records as a means of promoting efficiency. Some suggestions are offered as to the form in which such records may be kept. For those who desire to enter more fully into a study of commercial efficiency many useful books are now available.<sup>a</sup>

There is a tendency among many quarrymen to overestimate a loss of actual cash and underestimate a loss due to operating inefficiency, as the latter is less tangible. A mistake in bookkeeping by which a workman is paid \$2 for overtime that he did not put in is considered a serious matter. If, on the other hand, by channeling a cut instead of drilling and wedging it the same man increases the cost of production of a certain amount of marble by \$10, the mistake is overlooked. To the unobservant, heavy losses due to poor methods or machinery are classed as necessary burdens. Even the intelligent and observant operator may place them in the category of necessary working expenses unless a cost system reveals that they are avoidable.

The quarryman's first object should be to keep to a minimum the cost of each operation. It thus becomes his duty to continually improve his present methods, to devise more economical methods, or to maintain a low cost already realized. In order to do this, it has been stated that some efficient means must be employed for testing the various methods and determining accurately the relative value of each. A cost-keeping system supplies this need. It is a tool for cutting down costs.

Thus it may be seen that the highest efficiency demands careful technical records. It has been asserted by one economic writer that nine out of every ten failures in business are the direct result of a lack of proper knowledge of conditions, the deficiency being due to poor bookkeeping methods, or to none at all.

This is an age of competition. Manufacturers are straining every nerve to obtain contracts and to fill them at a profit. Many concerns have an uncomfortable feeling that progress and profits are not as satisfactory as they might be. Such concerns must realize that a cause must be definitely located and that the deficiency must be amended or other concerns will crowd them out. It is obvious that,

<sup>a</sup> See Church, A. H., *The proper distribution of expense burden*, 1913, 144 pp.; Church, A. H., *Production factors*, 1910, 187 pp.; Gantt, H. L., *Work, wages, and profits*, 1913, 312 pp.; Emerson, Harrington, *Efficiency*, 1912, 254 pp.; Emerson, Harrington, *The twelve principles of efficiency*, 1912, 423 pp.; and Carpenter, C. U., *Profit-making management*, 1908, 146 pp.



other things being equal, the business that is scientifically managed, that can put its finger on every item of excessive cost and immediately direct energy toward a reduction of that cost, is the business that will survive when others fail.

It is thought by many that the only function of a cost system is to ascertain the cost of an article. They completely overlook the fact that a proper analysis of costs furnishes a basis for reducing costs.

#### QUALIFICATIONS OF SUPERINTENDENTS.

Accounts are a measure of performance but must not be interpreted as performance itself. The establishment of a good system of technical accounting does not assume increased technical efficiency. There must be (1) the ability to interpret accounts and note wherein they reflect inefficiency, and (2) the knowledge and originality necessary to devise new methods and equipment. Such qualifications must be met not only by the general manager, but by the superintendents as well. They are closely in touch with all operations and control to a large extent the methods and equipment employed. The importance of these requisites demands some elaboration.

First, the superintendent must be master of his accounts. This statement does not mean that he must be a skilled accountant. As is shown later, the accounts involved are usually of a simple nature, easily kept and easily understood. Some may feel themselves handicapped by a lack of bookkeeping knowledge. It is a safe assumption, however, that it is much easier for the technical man to attain a sufficient mastery of accounts to interpret them in terms of efficiency than for the accountant to acquire technical knowledge. A premium should therefore be placed on foremen and superintendents who have in addition to their technical training a knowledge of mathematics and bookkeeping or the ability to assimilate the necessary knowledge of these subjects. That some operators employ foremen who can not read or write is almost incredible in this age of ever-increasing commercial efficiency, but it must be accepted as a fact. It is also a fact that some such quarries are not operating at a profit.

Second, the superintendent should have wide knowledge and originality. The ability to analyze and interpret costs is of limited value without a wide knowledge of ways and means of bettering conditions. He must know of methods and equipment used elsewhere or be able on occasion to devise absolutely new ones. The previous discussion of quarry and shop operation is designed to supply necessary information on methods and equipment now in use. Few superintendents can travel widely, but they should supplement their knowledge by visiting other quarries whenever possible.

### REAL PURPOSE OF TECHNICAL ACCOUNTS.

The duty of a technical accountant is not to find a convenient means of averaging and spreading costs over various operations on some arbitrary basis, but to separate and localize costs. His purpose is to enable him to follow with his cost book every operation from quarry to finished product and to see at a glance the cost of all the individual items that together constitute the total cost of production. He can then compare these costs with each other, observe which is the heaviest, and determine at what point his chief energies should be directed toward cost reduction. He can also compare the cost of the same operation through different periods of time. He can ascertain whether a new method increases or decreases the cost, and if a new machine is introduced he can determine whether it cuts down the cost of production in its limited field. Such considerations are of supreme importance. When selling prices are fixed by competition an increase in profit can be accomplished only by a decrease in cost of production. A general decrease can be achieved only by bringing about minor reductions here and there. If one is to know at once whether any new machine or method is contributing its share toward such reductions or is maintaining a minimum rate, cost keeping must be localized.

### THE COST OF NOT KEEPING ACCOUNTS.

The degree of localization in cost keeping must be governed by common sense. Bookkeeping can be carried to such an extreme that the extra expense of office help will eat up the profit from increased efficiency in other departments. However, reasonably detailed systems may be employed at little extra cost. Many quarrymen are opposed to them on account of the extra office help required. One quarryman remarked, "I can not afford to install a cost system." When reminded of the successful operation of the system in one quarry he replied, "Their margin of profit is so small that they have to keep close cost accounts." What sort of reasoning is it that leads to the conclusion that the one whose profit is presumably the larger can not afford a cost system, whereas the one whose margin of profit is narrow must burden himself with additional accountants? The two statements are in open conflict, and without doubt the second is correct. With all quarrymen the statement should be, "I can not afford to be without a cost system."

### LOCALIZATION OF COSTS.

The need of cost keeping having thus been pointed out, attention may be directed toward the principles governing the localization of control accounts. In order to properly analyze costs it is necessary

to divide them into the several factors that are more or less independent of each other. There are certain factors that constitute services toward production and yet are distinct from the actual and direct processes of production. The latter are labor and supplies directly applied to the object under construction, whereas the former are such items as power, rent, interest, supervision, and yard operation. The latter are necessary factors of production, but are less directly connected with the actual processes. The purpose of the technical accountant is to keep the expense figures of these various factors separate from each other so that the efficiency of each may be judged on its own individual merits and quite independent of the influence of any other operation.

Thus, if an operator considers scrapping his steam engine and purchasing electricity at  $1\frac{1}{2}$  cents per kilowatt-hour, if he has no records of the present cost of his power per horsepower-hour he has no means of knowing whether the change would be profitable. If, however, he has careful records of the cost of fuel, supplies, repairs, labor, maintenance of buildings, depreciation, interest on capital, etc., for his power plant and thus is able to fix for every month the cost of power per horsepower-hour he is in a position to judge whether a change is justifiable. A special account should be kept for each of the independent factors, which together make up the total cost of production. The total of each of the indirect factors must be apportioned on a fair basis between the different units comprising the entire plant, that is, the production centers.

#### GENERAL AND SPECIAL ACCOUNTS.

The records kept may be classed as general and special. General accounts are of two types: The first covers operations affecting the whole plant, and the items must on this account be apportioned among the various centers of production, such as quarry, mill, and shop; the second type consists of summarized statements of the total direct and overhead expense confined to each production center independent of all others.

The method of obtaining costs for each production center is shown in the following table, where quarry, mill, and shop are represented as production centers. The total cost for each production center is the actual direct and overhead expense of each, together with the proper proportion of general expense, yard expense, and power expense for each.



*Form for keeping costs of various production centers.*

Quarry.		Mill.		Shop.	
Item.	Cost.	Item.	Cost.	Item.	Cost.
Direct expense.....	\$.....	Direct expense.....	\$.....	Direct expense.....	\$.....
Overhead expense.....		Overhead expense.....		Overhead expense.....	
Proportion of general ex- pense allotted to quarry.....		Proportion of general expense allotted to mill.....		Proportion of general expense allotted to shop.....	
Proportion of yard ex- pense allotted to quarry.....		Proportion of yard expense allotted to mill.....		Proportion of yard expense allotted to shop.....	
Proportion of power ex- pense allotted to quarry.....		Proportion of power expense allotted to mill.....		Proportion of power expense allotted to shop.....	
Total, quarry.....		Total, mill.....		Total, shop.....	

Other general accounts may be kept showing total production and summarized statements of total and unit costs of each operation.

In recording all general accounts it is well for purposes of comparison to place in a parallel column the cost of each item for a preceding period of possibly six months. The average figure for the period would probably be the most useful, as it would show whether an item for any given month was below or above the average cost for the preceding period.

Special accounts are localized to each special operation. Their purpose is to show the working efficiency and the time efficiency of various machines or methods. By means of careful records useful comparisons may be made. In making comparative tests of two or more machines it is wise to employ the same operator on each machine during successive periods, as the efficiency of the operators may vary more than that of the machines. This suggests that special accounts may also be kept to test the relative efficiency of men.

#### GENERAL ACCOUNTS AFFECTING THE WHOLE PLANT.

The general accounts <sup>a</sup> which must be apportioned among the various production centers are (1) general expense, (2) cost of yard operation, (3) cost of power.

#### GENERAL EXPENSE.

The general-expense account is made up of all items which can not be directly applied to any single production center, such as quarry, mill, or shop. It consists of the land factor and the organization factor.

<sup>a</sup> The writer is indebted to Maj. J. S. Sewell for many suggestions relating to quarry accounts.

## LAND FACTOR.

When land is acquired for quarrying purposes either a certain amount of capital is tied up by purchase or rent must be paid. The interest or rent, together with taxes, is an annual fixed amount which must be charged against all operations conducted on the property.

## ORGANIZATION FACTOR.

Costs due to organization include expenses of administration, office, advertising, and expenses of a general nature which affect the plant as a whole.

The various items of general expense are totaled at the end of each month and apportioned to the various production centers on a fixed basis, as, for example, five-eighths to quarry, one-fourth to mill, and one-eighth to shop, or on whatever other fractional basis may seem justified. A general-expense account may have some such form as the following:

*Form of general-expense account.*

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	Dollars.
Administration.....		
Office expense.....		
Legal expense.....		
Traveling expense.....		
Interest and discount expense.....		
Advertising.....		
Taxes.....		
Insurance, fire.....		
Insurance, liability, etc.....		
Total.....		
Charge to quarry.....		
Charge to mill.....		
Charge to shop.....		

## YARD-OPERATION FACTOR.

The yard account includes interest on capital expended on yard equipment, wages, costs of supplies, maintenance and depreciation charges, and all other incidental expenses connected with transportation of rock from the quarry to the stock pile and from there to the mill and shop, or directly from the quarry to the mill and shop. It is a very variable account and may be omitted where the rock is loaded directly upon the transfer car with the quarry derrick. The following is a convenient form of account:

*Form of account for keeping yard costs.*

Item.	June.	
	Month.	Average for 6-month period.
	<i>Dollars.</i>	<i>Dollars.</i>
Superintendent.....		
Engineers and firemen.....		
Tracks:		
Labor.....		
Supplies.....		
Repairs to locomotive or crane:		
Labor.....		
Supplies.....		
Fuel.....		
Cars:		
Labor.....		
Supplies.....		
Miscellaneous.....		
Depreciation.....		
Total.....		
Charge to quarry.....		
Charge to mill.....		
Charge to shop.....		

## POWER FACTOR.

The simplest case in power charges is where power is purchased at so much per kilowatt-hour or horsepower. The next simplest case is where the transmission and distribution is done by the purchasing company; this will involve charges for interest on capital and for depreciation and maintenance of necessary equipment in addition to the first cost. Where steam power is used, or electricity is developed by steam, there will be the interest on capital expended in boilers, engines, generators, buildings, etc., in addition to charges for maintenance, labor, fuel, and supplies. In the case of hydroelectric plants the price of fuel is usually offset by the capital invested in dams and sluices.

It is important that exact power costs be kept and wherever possible reduced to the unit cost, or rate per kilowatt-hour or horsepower-hour. A large part of the loss connected with stone quarries is in the power factor. A power cost that is excessive as compared with the numerous published statistics of power-development costs indicates that something is radically wrong and that the services of an expert are required to stop the leak. If power from more than one plant is employed a separate account should be kept for each plant in order that the expense be properly localized. The power cost should be apportioned among the various production centers on the basis of the power actually consumed by each.

When power is purchased no account is necessary except for transmission and distribution. Power plants are of various types and the form of account must be varied to suit the conditions; two typical forms are given below.



*Form for costs of steam power plant.*

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	Dollars.
Firemen and helpers.....		
Engineers.....		
Maintenance of buildings:		
Labor.....		
Supplies.....		
Maintenance of boilers and engines:		
Labor.....		
Supplies.....		
Maintenance of steam lines:		
Labor.....		
Supplies.....		
Fuel.....		
Removing cinders:		
Labor.....		
Supplies.....		
Miscellaneous.....		
Superintendence.....		
Depreciation.....		
Total.....		
Cost per unit.....		

In the case where steam is conducted directly to the quarry machinery the total cost may be apportioned to the various machines in the special accounts for these machines. Where part of the steam power is used to develop electricity a separate account must be kept for this item.

*Cost form for hydroelectric power plant.*

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	Dollars.
Attendants.....		
Maintenance of buildings:		
Labor.....		
Supplies.....		
Maintenance of dams and sluices:		
Labor.....		
Supplies.....		
Maintenance of power lines:		
Labor.....		
Supplies.....		
Maintenance of turbines and generators:		
Labor.....		
Supplies.....		
Supplies (oil, waste, etc.).....		
Depreciation (amortization).....		
Total.....		
Cost per kilowatt-hour.....		
Charge to quarry.....		
Charge to mill.....		
Charge to shop.....		

This account gives the proportion of power expense for each department and also indicates the efficiency of the power plant. The cost of developing electric power depends on the amount of

capital invested in dams and machinery and on the length of transmission lines and hence is subject to considerable variation. It varies generally from  $1\frac{1}{2}$  to 2 cents per kilowatt-hour.

### GENERAL QUARRY ACCOUNTS.

The general quarry accounts are (1) direct quarry cost, (2) overhead expense, (3) total cost of operating quarry, (4) quarry-production account, and (5) condensed cost account.

### DIRECT QUARRY COST.

The direct quarry cost includes the labor and supplies required for each quarry operation. A form for keeping this account is given below:

*Form for keeping direct cost accounts for quarry.*

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	Dollars.
<b>Labor:</b>		
Channeling.....		
Setting up track.....		
Gadding.....		
Setting up gadders.....		
Raising blocks.....		
Pumping and cleaning up quarry.....		
Changing channeling machines.....		
Steel for channelers.....		
Steel for gadders.....		
Miscellaneous.....		
<b>Supplies:</b>		
Channeling.....		
Setting up track.....		
Gadding.....		
Setting up gadders.....		
Raising blocks.....		
Pumping and cleaning up.....		
Changing channeling machines.....		
Steel for channelers.....		
Steel for gadders.....		
Miscellaneous.....		
Fuel for heating.....		
Total direct quarry cost.....		

### OVERHEAD EXPENSE.

The distinction between overhead and general expense is clearly defined. The latter is made up of those items which affect the plant as a whole and which must be shared by each production center proportionally. The former is a separate account kept for each production center and includes for each center indirect expenses which pertain to that center only and are independent of all others. The ordinary expenses which fall in this class are maintenance and repairs and supervision—that is, the wages of superintendents and foremen employed in this particular center of production. It seems convenient also to include depreciation charges with overhead

expenses. Depreciation is an indirect expense and is an independent item for each production center. It is usually expressed as a percentage of the original cost calculated on a basis of the estimated time which may elapse before the equipment must be replaced. It usually varies within limits of 5 to 10 per cent per annum.

*Form for overhead-expense account for quarry.*

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	Dollars.
Superintendent and foremen .....		
Watchman .....		
Depreciation .....		
Labor:		
Maintenance of channelers .....		
Maintenance of gadders .....		
Maintenance of derricks .....		
Maintenance of pumps .....		
Maintenance of buildings .....		
Miscellaneous .....		
Supplies:		
Maintenance of channelers .....		
Maintenance of gadders .....		
Maintenance of derricks .....		
Maintenance of pumps .....		
Maintenance of buildings .....		
Miscellaneous .....		
Total overhead expense for quarry .....		

**TOTAL COST OF OPERATING QUARRY.**

The total quarry-cost account contains the various items which make up the total cost of operating the quarry. It includes the proportional share of general expense, yard, and power charges, and the total direct and overhead charges for the quarry.

*Form for keeping total quarry cost.*

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	Dollars.
Apportioned share of general expense .....		
Apportioned share of yard service .....		
Apportioned share of power .....		
Total direct operating cost .....		
Total quarry overhead cost .....		
Total cost of operating quarry .....		

Circumstances may make it advisable to keep other accounts, such as a special development account.



## QUARRY PRODUCTION ACCOUNT.

Mere statements of costs are of little value unless accompanied by a statement of production for the same period.

*Form for keeping quarry production account.*

Item.	June,	
	Month.	Average for 6-month period.
	<i>Cubic feet.</i>	<i>Cubic feet.</i>
Gross production.....		
Scrap blocks.....		
Net production.....		
Quantity delivered to mill (or shipped wholesale).....		
Quantity delivered to yard.....		
Quantity delivered from yard to mill (or shipped wholesale).....		

## CONDENSED COST ACCOUNT.

A summarized account may be kept of the total and unit costs of each operation, including the overhead charge. It is compiled from other accounts, and presents the facts in a convenient form for reference or comparison. A convenient form for keeping such an account is given below:

*Form for keeping condensed cost account.*

Item.	June,	
	Month.	Average for 6-month period.
	<i>Dollars.</i>	<i>Dollars.</i>
Total cost of channeling (includes quarry overhead charge).....		
Total number of square feet channeled.....		
Cost of channeling, per square foot.....		
Vertical drilling:		
Total cost (including overhead charge).....		
Total number of square feet of rock drilled.....		
Total number of linear feet drilled.....		
Cost per square foot.....		
Cost per linear foot.....		
Bed drilling: <sup>a</sup>		
Total cost (including quarry overhead charge).....		
Total number of square feet of rock drilled.....		
Total number of linear feet drilled.....		
Cost per square foot.....		
Cost per linear foot.....		
Total cost of lifting blocks (including quarry overhead charge).....		
Total cost of maintaining quarry.....		
Total cost of power.....		
Total cost of quarry plant depreciation.....		
Total cost of development work.....		

<sup>a</sup>A term applied to drill holes made for the purpose of freeing the block from the floor.

## SPECIAL QUARRY ACCOUNTS.

The accounts just considered cover the total quarry cost, the total production, and the total and unit costs of each operation. More detailed records are necessary, however, both for the purpose of compiling the general records, and to check the efficiency of each separate operation.

In order to test channeling or drilling efficiency no account is taken of the proportion of waste, for it makes no difference whether sound or unsound material is produced, as regards the rate of cutting. Hence all such records are based on gross production. For channeling the following records should be kept:

*Form for keeping channeling costs, based on gross production.*

Item.	Month.
Square feet channeled.....	.....
Channeling cost.....	.....
Square feet cut in 10-hour day.....	.....
Square feet cut in 10-hours' cutting time.....	.....
Channeling cost per square foot.....	.....
Square feet channeled per cubic foot.....	.....
Channeling cost per cubic foot.....	.....

*Form for channeling time-efficiency account.*

Time.	Month.
Maximum working hours.....	.....
Cutting time, per cent.....	.....
Time used for repairs, per cent.....	.....
Time used in moving and setting track, per cent.....	.....

The following gadding accounts should be kept:

*Account for keeping gadding costs, based on gross production.*

Item.	Month.
Square feet of gadding.....	.....
Gadding cost.....	.....
Square feet cut in 10-hour day.....	.....
Square feet cut in 10 hours' cutting time.....	.....
Gadding cost per square foot.....	.....
Square feet of gadding per cubic foot.....	.....
Gadding cost per cubic foot.....	.....
Linear feet cut.....	.....
Cost per linear foot.....	.....

*Account for keeping gadding time efficiency.*

Time.	Month.
Maximum working hours.....	.....
Cutting time, per cent.....	.....
Time used for repairs, per cent.....	.....
Time used for moving, per cent.....	.....

Separate accounts may be kept for vertical drilling and bed drilling, or they may be united. Both channeling and gadding accounts may for purposes of comparison be localized to machines of the same class or even to individual machines.

Such accounts also afford a means of testing various methods. Suppose a change is made in the direction of a row of drill holes so that advantage is taken of a rift in the rock which permits a wider spacing of the holes. The result would be shown in the decreased gadding cost per cubic foot of rock produced. The exact saving thus effected may be balanced against any increased cost which may be involved in the change of quarry method, and thus the operator will know whether such a plan is justified.

As a further illustration, suppose the rift to be so poorly defined that it may seem advisable to channel in two directions and gad only on the bed. Such a change would lower the gadding cost and increase the channeling cost per cubic foot of rock produced. The sum of the channeling and gadding costs before the change compared with their sum after the change would indicate the loss or gain involved in such a change of method.

The preceding statements indicate that in many cases conclusions must not be drawn from isolated accounts, but from the relation of one account to another. The accounts are often interdependent, and no one can interpret them properly except the man who is familiar with both the quarry operations and the accounts kept.

A very useful account is one which summarizes the unit cost of each item on a basis of gross production followed by the same item on a basis of net production, as follows:

*Account for keeping gross and net unit costs.*

Item.	June.			
	Month.		Average for 6-month period.	
	Gross.	Net.	Gross.	Net.
	Dollars.	Dollars.	Dollars.	Dollars.
Channeling, per cubic foot.....				
Vertical drilling, per cubic foot.....				
Bed drilling, per cubic foot.....				
Lifting blocks, per cubic foot.....				
Maintenance of quarry, per cubic foot produced.....				
Power, per cubic foot produced.....				
Total cost.....				

Such an account indicates clearly how the proportion of waste adds to the cost of production of the finished product.

For a certain month the total cost of production per cubic foot, gross measure, in one marble quarry, was only 35 cents. When calculated



on the basis of net production it was, however, \$1.31 per cubic foot. This means that if all the marble produced were salable—an ideal condition which is impossible in practice but may be assumed for purposes of illustration—the marble could be sold at 36 cents a cubic foot for a profit, if selling expenses were disregarded. So much rock is thrown away on the waste heap, however, that with no compensating revenue from waste material the good stock must be sold for more than \$1.31 in order to realize a profit. If more quarrymen kept records which would clearly indicate the excessive losses that result from quarrying inferior material, it is safe to assume that more active steps would be taken in an attempt to modify quarry methods in such a manner as to reduce the proportion of waste.

Block records are very useful, as they assist the quarry operator in keeping track of his product. A form of record kept by one marble company is as follows:

*Block record kept by a North Carolina company.*

No. of block.	Date quarried.	Dimensions.	Grade.	Total cubic feet.	To mill.
1243.....	July 7, 1914	3' 0'' by 3' 0'' by 1' 0''.	No. 3	9	August 4, 1914
.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....

For companies who sell their stock wholesale a block record is still more desirable and requires greater detail. A form used by a Tennessee marble company, which is very complete and enables the operator to trace every block from the quarry pit to its final destination, is given below:

*Form of block record kept by wholesale company.*

Datelistd.	List No.	Grade.	Color.	Official measure.	Cubic feet.	Accepted measure.
May 10	1234	1	Pink	7' 0'' by 6' 0'' by 3' 0''.	126	.....
May 10	1235	1	...do...	6' 6'' by 6' 0'' by 3' 0''.	117	.....
.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....



*Form for keeping direct mill costs.*

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	Dollars.
<b>Labor:</b>		
Loading and unloading gangs.....		
Sawing.....		
Hoisting engineer.....		
Placing stock.....		
Sawing with diamond saw.....		
Removing yard scrap and waste.....		
Loading sawed stock in cars.....		
Bracing sawed stock in cars.....		
Water supply.....		
<b>Supplies:</b>		
For loading and unloading gangs.....		
For hoisting engineer.....		
For placing stock.....		
For removing yard scrap and waste.....		
For loading sawed stock in cars.....		
For bracing sawed stock in cars.....		
Saw blades.....		
Sand.....		
Total direct mill cost.....		

## OVERHEAD EXPENSE.

Overhead expense includes cost of superintendence, maintenance and repairs, and depreciation. A form for keeping an overhead-expense account for the mill is given below.

*Form of overhead-expense account for mill.*

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	Dollars.
Superintendent.....		
Foreman.....		
Depreciation.....		
<b>Labor:</b>		
Maintenance of gangs and sand pumps.....		
Maintenance of diamond saws.....		
Maintenance of crane and runway.....		
Maintenance of shafting and belting.....		
Maintenance of drain and settling basin.....		
Maintenance of buildings.....		
Maintenance of machine shop and equipment.....		
Maintenance of heating.....		
Watchman.....		
Miscellaneous.....		
<b>Supplies:</b>		
Maintenance of gangs and sand pumps.....		
Maintenance of diamond saws.....		
Maintenance of crane and runway.....		
Maintenance of shafting and belting.....		
Maintenance of drain and settling basin.....		
Maintenance of buildings.....		
Maintenance of machine shop and equipment.....		
For heating.....		
For watchman.....		
Miscellaneous.....		
Total overhead.....		



## TOTAL COST OF OPERATING MILL.

A form for keeping a summarized statement of mill costs, similar to that for the quarry, is given below:

*Form for keeping total costs of operating mill.*

Item.	June.	
	Month.	Average for 6-month period.
	<i>Dollars.</i>	<i>Dollars.</i>
Apportioned share of general expense.....		
Apportioned share of yard service.....		
Apportioned share of power.....		
Total direct operating cost.....		
Total overhead charge for mill.....		
Total cost of operating mill.....		

## UNIT COST ACCOUNT.

The unit cost account shows the amount of material produced and the unit costs direct and indirect for gangs only. If diamond saws or wire saws are employed, separate accounts should be kept for them.

*Form of unit cost account for gangs.*

Item.	June.	
	Month.	Average for 6-month period.
	<i>Dollars.</i>	<i>Dollars.</i>
Number of gang hours actually made.....		
Direct cost per gang hour.....		
Total cost per gang hour.....		
Square feet of saw cuts.....		
Direct cost per square foot of saw cut.....		
Total cost per square foot of saw cut.....		
Cubic feet of blocks for first cut.....		
Cubic feet of blocks for second and third cuts.....		
Cubic feet of blocks for all stock loaded.....		
Total cubic feet sawed (all between saws).....		
Total cubic feet good color obtained.....		
Total cubic feet good and sound.....		
Direct cost per cubic foot total sawed.....		
Total cost per cubic foot total sawed.....		
Direct cost per cubic foot good color.....		
Total cost per cubic foot good color.....		
Direct cost per cubic foot good and sound.....		
Total cost per cubic foot good and sound.....		
Average square feet saw cuts per gang hour.....		
Average square feet sawed per gang hour.....		
Rate of sawing.....		
Average number of blades per gang load.....		
Average length of stock.....		
Average number of cubic feet per gang load.....		
Number of gang loads.....		

This form of account is subject to many modifications. The one given is designed to fit a complex condition, where the marble is marred by both bad color and unsoundness. If either of these is absent the account may be simplified. If the material is graded as No. 1, 2, and 3 as it leaves the saws the account must be modified to suit.

## CONDENSED COST ACCOUNT.

The value of material as it enters the mill is the sum of the costs of work already done upon it as indicated by the quarry records. The summarized account gives in condensed form the cost of material through the mill and indicates the additional costs of various operations, both as totals and units.

*Form for keeping condensed cost account.*

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	Dollars.
Value of blocks to mill from quarry.....		
Value of blocks to mill from yard.....		
(a) Total value of blocks to mill.....		
Average value per cubic foot of blocks to mill.....		
(b) Total mill operating cost.....		
Total value of output of mill (sum of a and b).....		
Thin stock: <sup>a</sup>		
Value of blocks delivered to mill.....		
Total cost of sawing.....		
Total value of sawed stock produced.....		
Average value of sawed stock produced.....		

<sup>a</sup> Similar accounts may be kept to ascertain the costs per cubic foot of making the first, second, and third cuts, cuts subsequent to the third, and of cutting with the diamond saw.

For convenience in keeping saw-gang records, the accompanying ticket form which is used by one marble company may offer useful suggestions:

## SAW TICKET.

Gang No. .... Dimensions ..... Block No. .... Grade.....

	Date.	Hour.	
When last down.....			No. saws.....
When started.....			No. pes.....
When down.....			

Total hours under gang..... Idle for loading..... Sizes sawed into.....

## HOURS RUN AND INCHES SAWED EACH SHIFT.

Sawyer.	1		3		5		7	
	Hrs.	Ins.	Hrs.	Ins.	Hrs.	Ins.	Hrs.	Ins.
Totals.....								

Total hours sawed ..... Total inches sawed ..... Total hours delay .....

## DELAYS.

Date.	Hours.	Cause.
.....	.....	.....
.....	.....	.....
.....	.....	.....
.....	.....	.....

## SPECIAL MILL ACCOUNTS.

The special mill accounts show the amounts produced by the various operations and the total and unit costs of each operation, with certain other useful figures such as rate of sawing, number of blades, etc. A summarized statement of total quantities produced and cost of the same may be given as follows:

*Form for keeping cost account of total quantities produced.*

Total quantities produced by gang.	June.	
	Month.	Average for 6-month period.
Gang hours actually made.....	.....	.....
Direct cost per gang hour.....	.....	.....
Total cost per gang hour.....	.....	.....
Square feet of saw cuts.....	.....	.....
Direct cost per square foot of saw cut.....	.....	.....
Total cost per square foot of saw cut.....	.....	.....
Gross measure blocks for first cut.....	.....	.....
Gross measure blocks for second and third cuts.....	.....	.....
Gross measure all stock loaded.....	.....	.....
Total <sup>a</sup> cubic feet sawed (all between saws).....	.....	.....
Direct cost per cubic foot (total sawed).....	.....	.....
Total cost per cubic foot (total sawed).....	.....	.....
Average number of square feet sawed per gang hour.....	.....	.....
Average number of cubic feet sawed per gang hour.....	.....	.....
Rate of sawing.....	.....	.....
Average number of blades per gang load.....	.....	.....
Average length of stock.....	.....	.....
Average number of cubic feet per gang load.....	.....	.....
Number of gang loads.....	.....	.....

<sup>a</sup> Similar items showing the cost of producing the different grades may be given if desired.

A thin-stock account may be kept in more or less detail. The form given here may be modified at the operator's discretion.



*Form for keeping thin-stock account.*

Thin stock.	June.	
	Month.	Average for 6-month period.
Gross measure, cubic feet.....		
Total cubic feet sawed (all between saws).....		
Total number of square feet of saw cuts.....		
Total gang hours.....		
Direct operating cost.....		
Total operating cost.....		
Direct cost per cubic foot of total quantity sawed.....		
Total cost per cubic foot of total quantity sawed.....		
Direct cost per square foot of saw cut.....		
Total cost per square foot of saw cut.....		
Average number of square feet sawed per gang hour.....		
Average number of cubic feet cut per gang hour.....		
Average rate of sawing.....		
Average number of blades per gang load.....		
Average length of stock.....		
Average number of cubic feet per gang load.....		
Total number of gang loads.....		

<sup>a</sup> Items similar to this one for the different grades might be inserted here.

Similar accounts may be kept for cubic stock, showing the cost of making the first cut, second cut, third cut, and cuts subsequent to the third cut, and for diamond saws. Cuts subsequent to the third cut are usually diagonal or some such direction for the purpose of decreasing mill labor, and as a consequence are charged to the mill.

Time-efficiency reports are of great value. The various causes of delay may be enumerated and calculated to percentages of the maximum sawing hours, as in the form following:

*Form for time-efficiency record of gang.*

Item.	June.			
	Month.		Average for 6-month period.	
	Hours.	per cent.	Hours.	per cent.
Maximum sawing time.....				
Actual sawing time.....				
Time consumed by delays:				
Loading and unloading.....				
Coming down at night.....				
Without water.....				
Without power.....				
Repairs to gangs.....				
Repairs to belting and shafting.....				
Repairs to pumps.....				
Repairs to lead pipes.....				
Without blocks.....				

Suppose that in addition to a traveling crane a transfer-car system is installed. In the next monthly report the advantage of this additional equipment would be reflected in the time-efficiency record as a material reduction in the time consumed in loading and unloading.

**SHOP OR FINISHING PLANT ACCOUNTS.**

In considering the various operations in the shop a new condition presents itself. Although the whole shop has been termed a production center it is in reality a group of smaller independent production centers comprising coping, rubbing plain stock, gritting plain stock, buffing plain stock, cutting, matching, drilling holes for anchors, boxing, and loading. Keeping in mind the principle that technical accounts should be localized if costs are to be properly isolated, for each of these smaller production centers, a separate account must be kept, each bearing its proper share of general expense, yard service and power, and its total direct and overhead charge. The apportioning of general expense and other indirect costs requires careful adjustment. Some departments require more supervision or more yard service than others, and each should be burdened with no more than its fair share. Power cost is apportioned on the basis of the actual amount of power used by each department.

Two general shop accounts are necessary, one being a summary of shop costs and the other a summary of shop production.

*Cost account for shop or finishing plant.*

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	Dollars.
General expense:		
Percentage of main general expense account.....		
Draftsmen (percentage of total).....		
Drafting supplies (percentage of total).....		
Total general expenses.....		
Yard operation (percentage of total).....		
Power (amount actually used).....		
Direct shop costs:		
Labor.....		
Supplies.....		
Total direct shop costs.....		
Overhead expense for shop:		
Draftsmen (percentage of total).....		
Drafting supplies (percentage of total).....		
Superintendence.....		
Shop depreciation.....		
Maintenance of shop machinery—		
Labor.....		
Supplies.....		
Total overhead expense for shop.....		
Summary:		
General expenses.....		
Yard operation.....		
Power.....		
Direct shop costs.....		
Overhead expense for shop.....		
Total cost of operating shop.....		
Total number of productive hours of shop labor.....		
General expense per productive hour.....		
Yard expense per productive hour.....		
Power expense per productive hour.....		
Direct expense per productive hour.....		
Overhead expense per productive hour.....		

A plain-stock production account summarizes the number of cubic feet of stock handled and the work done on it at each operation.

*Form for keeping records of plain-stock production.*

Plain-stock production.	June.	
	Month.	Average for 6-month period.
Square feet of stock coped (excluding tile).....		
Cubic feet of stock coped (excluding tile).....		
Square feet of tile coped.....		
Cubic feet of tile coped.....		
Square feet of stock rubbed (excluding tile).....		
Square feet of second surface rubbed (excluding tile).....		
Total square feet rubbed.....		
Cubic feet of stock rubbed.....		
Square feet of tile rubbed.....		
Cubic feet of tile rubbed.....		
Square feet of stock gritted.....		
Square feet of second surface gritted.....		
Total square feet gritted.....		
Cubic feet of stock gritted.....		
Square feet of stock buffed.....		
Square feet of second surface buffed.....		
Total square feet buffed.....		
Cubic feet of stock buffed.....		

The account for each of the production centers within the shop may have somewhat the following form:

*Form for keeping costs of individual operations.*

Coping.	June.	
	Month.	Average for 6-month period.
Total number of productive hours.....		
Square feet of stock coped.....		
Cubic feet of stock coped.....		
General expense.....		
Yard service.....		
Power.....		
Overhead shop expense.....		
Direct expense:		
Copers.....		
Helpers.....		
Foremen.....		
Removing scrap.....		
Supplies.....		
Total cost of coping per cubic foot ( <i>a</i> ).....		
Cost of stock per cubic foot before coping ( <i>b</i> ).....		
Value of coped stock per cubic foot (sum of <i>a</i> and <i>b</i> ).....		

In this account the value of the stock is the sum total of the costs of all previous operations. After it has been coped the stock will have increased in value by the amount of the coping costs. The same process is followed during all subsequent operations, and the final boxing and loading account will show as the value of the stock the total cost of all operations through quarry, mill, and shop. The



accounts for rubbing plain stock, gritting plain stock, buffing plain stock, cutting, matching, and other necessary processes are kept in the same form as the coping account, though certain modifications must be made in the direct expense of each. Tile stock or refinishing may require additional accounts.

Accounts should be kept for all by-products, such as terrazzo, road material, fertilizer, riprap, lime, etc.

### JOB ACCOUNTS.

In shops where novelties are manufactured it may seem desirable to determine the total cost of an article which has passed through the hands of a number of operators. Such a record when placed on file affords a ready means of making future estimates on similar articles. If a block of cubic stock is taken, its first cost may easily be ascertained from the total cost of third-cut cubic stock in the condensed cost account of the mill records. The final cost will be this amount plus the cost of all subsequent work done upon it. For keeping such records the following forms of cost tickets are suggested. When the job is completed, the information contained in ticket 1 is transferred in condensed form to ticket 2. The latter may be of cardboard, of convenient size and shape for filing.

#### *Suggested forms for cost tickets.*

##### COST TICKET NO. 1.

Order No. .... Design.....  
 Kind of marble..... Size .....

Article.....

Class of workman.	Name of workman.	Time started.	Time stopped.	Total time.	Rate.	Cost.	Remarks.
Cutter.....	L. Smith..	7.30 a. m. 6/27/14	11 a. m. 6/27/14	3½ hrs.	50¢	\$1.75	
Cutter.....							
Cutter.....							
Polisher.....							
Polisher.....							
Tracer.....							
Turner.....							
Lathe polisher.....							

##### COST TICKET NO. 2.

Design..... Order No. .... For..... Price .....

Sizes.	Cubic feet.	Cost of—					Total cost.
		Cutting.	Polishing.	Tracing.	Turning.	Lettering.	

## PUBLICATIONS ON MINERAL TECHNOLOGY.

A limited supply of the following publications of the Bureau of Mines is temporarily available for free distribution. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

BULLETIN 16.—The uses of peat for fuel and other purposes, by C. A. Davis. 1911. 214 pp., 1 pl., 1 fig.

BULLETIN 42.—The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs.

BULLETIN 45.—Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 47.—Notes on mineral wastes, by C. L. Parsons. 1912. 44 pp.

BULLETIN 53.—Mining and treatment of feldspar and kaolin in the southern Appalachian region, by A. S. Watts. 1913. 170 pp., 16 pls., 12 figs.

BULLETIN 64.—The titaniferous iron ores of the United States, their composition and economic value, by J. T. Singewald, jr. 1913. 145 pp., 16 pls., 3 figs.

BULLETIN 81.—The smelting of copper ores in the electric furnace, by D. A. Lyon and R. M. Keeney. 1915. 80 pp., 6 figs.

BULLETIN 84.—Metallurgical smoke, by C. H. Fulton. 1915. 94 pp., 6 pls., 15 figs.

TECHNICAL PAPER 3.—Specifications for the purchase of fuel oil for the Government, with directions for sampling oil and natural gas, by I. C. Allen. 1911. 13 pp.

TECHNICAL PAPER 8.—Methods of analyzing coal and coke, by F. M. Stanton and A. C. Fieldner. 1913. 42 pp., 12 figs.

TECHNICAL PAPER 14.—Apparatus for gas-analysis laboratories at coal mines, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 7 figs.

TECHNICAL PAPER 32.—The cementing process of excluding water from oil wells, as practised in California, by Ralph Arnold and V. R. Garfias. 1912. 12 pp., 1 fig.

TECHNICAL PAPER 38.—Wastes in the production and utilization of natural gas, and means for their prevention, by Ralph Arnold and F. G. Clapp. 1913. 29 pp.

TECHNICAL PAPER 39.—The inflammable gases in mine air, by G. A. Burrell and F. M. Seibert. 24 pp., 2 figs.

TECHNICAL PAPER 41.—Mining and treatment of lead and zinc ores in the Joplin district, Missouri, a preliminary report, by C. A. Wright. 1913. 43 pp., 5 figs.

TECHNICAL PAPER 43.—The influence of inert gases on inflammable gaseous mixtures, by J. K. Clement. 1913. 24 pp., 1 pl., 8 figs.

TECHNICAL PAPER 50.—Metallurgical coke, by A. W. Belden. 1913. 48 pp., 1 pl., 23 figs.

TECHNICAL PAPER 66.—Mud-laden fluid applied to well drilling, by J. A. Pollard and A. G. Heggem. 1914. 21 pp., 12 figs.

TECHNICAL PAPER 68.—Drilling wells in Oklahoma by the mud-laden fluid method, by A. G. Heggem and J. A. Pollard. 1914. 27 pp., 5 figs.

TECHNICAL PAPER 70.—Methods of oil recovery in California, by Ralph Arnold and V. R. Garfias. 1914. 57 pp., 7 figs.

TECHNICAL PAPER 76.—Notes on the sampling and analysis of coal, by A. C. Fieldner. 1914. 59 pp., 6 figs.

TECHNICAL PAPER 88.—The radium-uranium ratio in carnotites, by S. C. Lind and C. F. Whittemore. 1915. 29 pp., 1 pl., 4 figs.

TECHNICAL PAPER 95.—Mining and milling of lead and zinc ores in the Wisconsin district, Wisconsin, by C. A. Wright. 1915. 39 pp., 2 pls., 5 figs.

TECHNICAL PAPER 110.—Monazite, thorium, and mesothorium, by Karl L. Kithil. 1915. 32 pp., 1 fig.

TECHNICAL PAPER 111.—Safety in stone quarrying, by Oliver Bowles. 1915. 48 pp., 5 pls., 4 figs.

TECHNICAL PAPER 126.—The casting of clay wares, by T. G. McDougall. 1916. 12 pp., 2 figs.

16207°—Bull. 106—16—12





# INDEX.

A.	Page.
Abrasion, causes of.....	8
resistance to.....	8
Abrasives for rubbing.....	103
for sawing.....	98-99
Absorption. <i>See</i> Porosity.	
Accounts. <i>See</i> Cost keeping.	
Adams, F. D., on elasticity in rock.....	133-134
Alabama marble, color of.....	11
Alabama quarries, methods used in.....	55,
65, 69, 74, 76, 85-86	
Allen, E. T., cited.....	30
Alumina in marble.....	6
Anticlines as evidences of compression.....	125,
128, 130, 131	
Appleton, Wis., anticlinal arch at.....	126, 127
B.	
Balusters, manufacture of.....	105
Beard, J. T., on mine explosions.....	128
Becker, G. F., on cause of cleavage.....	36
on joints.....	24, 25, 27
on surface tension of water in joints.....	23
Bedding, quarry operations relative to.....	56,
57, 58-61, 64-68, 79, 81, 109-110, 113-114, 115	
Berea, Ohio, rock pressure in quarries at.....	124
Blocks, acute-angled, quarrying of.....	114
waste due to.....	109
breaking of.....	120
irregular, waste in.....	111-112
"monoclinic," quarrying of.....	112
waste due to.....	115
"oblique," waste due to.....	110
regular, quarrying of.....	112-113
"right-angled," quarrying of.....	113-114, 115, 117
"triclinic," quarrying of.....	114-115, 116
<i>See also</i> Key blocks.	
"Boulder quarries".....	20, 69
Brunton, D. W., cited.....	50
Buckley, E. R., on determination of specific gravity.....	9
on effect of frost.....	38
on elasticity of marble.....	18
Buildings, marble for.....	20, 21
Burchard, E. F., on production of marble.....	22
By-products, quarry, manufacture of.....	119-120, 121
cost of.....	108
C.	
Calcite in marble, hardness of.....	8
solubility of.....	11
specific gravity of.....	9
Calcium carbonate, percentage in marble.....	5
Campbell, D. F., on weathering.....	128, 131
Carborundum, use of, for finishing.....	103-107
Carrara marble, translucence of.....	13
Cement, manufacture of.....	122
Chalmers, R., on thrust faults.....	127
Chamberlain, T. C., on earthquake strains.....	132
on recrystallization.....	133
Channeling machines, use of.....	53-54, 57-58, 61, 76
Channeling methods.....	53-71,
75-76, 79, 81, 85, 86, 87, 108, 111-117, 137, 143	

	Page.
Chert in marble.....	8
Chlorite in marble.....	6
Clarendon, Vt., strain breaks in quarry near.....	129
Cleavage, cause of.....	36
Cleopatra's needle, disintegration of.....	19
Cleveland, Ohio, anticlines in shales near.....	128
Clips for guys.....	90
Cockeysville, Md., quarries, fertilizer from.....	121
Coker, E. G. <i>See</i> Adams, F. D.	
Color bands, channeling with reference to.....	59
Color in marble, causes of.....	11-13
permanence of.....	35-36
varieties of.....	11-12
Colorado marble, color of.....	12
translucence of.....	13
Colorado quarries, methods in.....	72, 86
Colorado-Yule marble, color of.....	12
Columns, methods of cutting.....	106-107
Coping machine, use of.....	102
Cost, factors determining.....	145-146
Cost keeping, forms for.....	149-167
need of.....	144, 147-148
Cramer, Frank, on anticlinal uplift.....	126-127, 131
Crenshaw, J. L., cited.....	30
Crosby, W. O., on joints.....	24-25
Crosby clips for guys.....	90
Cross breaks, methods of making....	64, 69, 71, 73, 76

## D.

Dale, T. N., on colors of marble.....	12
on compression in granite quarries.....	128, 131
on disintegration of marble.....	19
on durability of marble.....	37
on grain of marble.....	14
on strain relief.....	139
on texture of Vermont marble.....	13-14
on translucence of foreign marbles.....	13
Daubrée, A., on joints.....	23
Davis, J. A., cited.....	50
Defreestville, N. Y., rock faults at.....	128
Dent Head, England, fracturing in quarry at.....	126
Derricks, guys for.....	90
placing of.....	91-92
size of.....	92
types of.....	89-90
Diopside in marble.....	6
Dip, determination of.....	41-43, 48, 49
influence of, on quarrying.....	56-61, 64-68, 109-110
Dolomite in marble, effect of.....	35, 118
hardness of.....	35, 118
solubility of.....	11
specific gravity of.....	6
Drainage of quarries.....	88
Dressing plant, fireproof buildings for.....	95
heating of.....	95
location of.....	94-95, 102
methods in.....	72, 101-108
plan of.....	97-98
power for.....	96, 151-153
water for.....	107-108
Drill cores, arrangement of.....	45, 76
determination of dip from.....	42

	Page.		Page.
Drill cores, preservation of .....	46	Hornblende in marble .....	6
value of .....	45	Hoskins, L. M., on earthquakes .....	133
Drill holes, for prospecting, arrangement of .....	42-45, 109	Hughes, T. McK., on rock flowage .....	131
depth of .....	42	on rock strains .....	126
for quarrying, arrangement of .....	75-81	I.	
reaming of .....	76-77	Iron oxides in marble .....	6
size of .....	75-76	Iron sulphides in marble .....	29-33
for relieving rock stresses .....	137-138	Italian breccias, color of .....	12
Drilling, cost of .....	74-75, 79, 143	J.	
rate of .....	74	Jackhammer drill, construction of .....	50
Drilling machinery, use of .....	74-75	operation of .....	74-75
Drills, bar .....	75	Johnson, John, on strains in sandstone .....	124
diamond .....	40	cited .....	30
double core-barrel .....	41, 109	Joint systems, arrangement of .....	23-26
hollow-steel .....	75	figure showing .....	26-27
jackhammer .....	74-75	influence of, on quarrying .....	61-69
sharpening of .....	77	Joints, arrangement of .....	23, 24
Dutton, C. E., on earthquakes .....	133	"fish-backs" .....	23
E.		causes of .....	23-26
Earthquakes, causes of .....	130-133	determination of .....	69
classes of .....	131	development of .....	25
types of .....	132	experimental study on .....	23-24
Efficiency, importance of .....	145-146	parallel .....	23, 26
Egleston, J., on decay of building stones .....	8	persistence of .....	27, 28, 29
on weathering of tombstones .....	8	recementing of .....	29
Electric air quarrying machines, use of .....	50, 114	surface tension of water in .....	23
Electricity as power, source of .....	51-52, 96	Julien, A. A., on decomposition of iron pyrites .....	31-32
use of .....	50-51, 96	on stability of sulphides .....	32
<i>See also</i> Power plants.		K.	
F.		Kasota, Minn., recrystallization of marble at .....	21
Fireproof buildings at marble works .....	95	Kaukauna, Wis., stresses in rocks near .....	126-127
"Fish-backs." <i>See</i> Joints.		Kennebec, Me., compression in quarries at .....	128
Fissility. <i>See</i> Cleavage.		Key blocks, removal of .....	80-82
Flint in marble .....	6	Knoxville, Tenn., methods at quarry near .....	68
Florence, Vt., strain breaks in quarry near .....	129	strains in quarry near .....	129
Flux, sale of scrap as .....	120	L.	
Folding of rocks .....	125	Lake Champlain region, marble, color of .....	12
G.		Lawson, A. C., on postglacial faults .....	129, 130, 131
Gang cars, loading of .....	99	Lee, Mass., composition of marble at .....	5
use of .....	98	Lemont, Ill., compression in quarries at .....	125
Gantry cranes, use of .....	90	Leschen clips for guys .....	90
Geologic maps, value of .....	39	Lewis pin, use of .....	82
Georgia quarries, methods in .....	48, 70-71, 75, 88	Lime, as fertilizer .....	121-122
Gilbert, G. K., on earthquakes .....	132	manufacture of .....	120
on expansion of rock .....	125-126	Lime Rock, N. Y., anticlines at .....	127
on postglacial folding .....	125, 127, 130	Limestone, composition of .....	5
Glass seams. <i>See</i> Joints, recementing of.		formation of .....	6
Grain. <i>See</i> Rift.		metamorphism of .....	7
Graphite in marble .....	6	Limonite in marble .....	6
Gresley, W. S., on fracturing of coal .....	126	Lincoln County, Me., compression in quarry in .....	128
Groton, Conn., compression in quarries at .....	125	Lithium in marble .....	6
Guys, arrangement of .....	90	Lithonia, Ga., anticlines near .....	130, 131
clips for .....	90	compression in quarry at .....	128
size of .....	90	M.	
H.		MacDonald, D. F., on earthquakes in Panama .....	132
Hancock, Vt., dolomite, color of .....	12	Magnesium carbonate in marble .....	5
Hancock County, Me., compression in quarries in .....	128	Manganese oxide in marble .....	6
Hardness of marble .....	8, 35	Marble, carving of .....	107
<i>See also</i> Calcite, Dolomite, Silica.		chemical impurities in .....	6
Haulage. <i>See</i> Transportation.		classes of .....	20, 21
Heating of quarry buildings .....	95	color of .....	11-13
Hematite in marble .....	6	permanence of .....	35-36
Hobbs, W. H., on earth's crust .....	131-132	composition of .....	5
Hoisting, methods of .....	91		
signals for .....	92		
Hooson, William, on subterranean noises .....	124		



	Page.
Marble, crushing strength of .....	17
definition of .....	5
disintegration of, agents influencing.....	18-19
rate of .....	18-20, 37
dressing of, plants for .....	94
elasticity of .....	17-18
foreign .....	13
in United States, distribution of .....	21, 22
iron sulphides in, effect of .....	30-31, 33
jointing of .....	22-29
mineral impurities in .....	
origin of .....	6
permeability of .....	33
porosity of, determination of .....	14-16, 37-38
prospecting for .....	39-40, 41
cost of .....	45
resistance to abrasion of .....	8
shearing of .....	18
shipping of .....	107
solubility of .....	11, 37
specific gravity of .....	9-10
texture of .....	13, 37
influence of impurities on .....	14
translucence of .....	13
unsoundness of, causes of .....	22-29
weathering of .....	20
weight of .....	10-11
<i>See also</i> Onyx, Quarries, Quarrying, Verd antique.	
Marble cutting, methods of .....	59-65
66, 68-80, 81, 84, 86, 87-88,	
102, 104, 106, 107, 111-117, 136	
Marble polishing.....	103-104, 106
by wind .....	8
Marcasite in marble .....	6
composition of .....	30
decomposition of, effect of .....	30, 31, 32, 118
hardness of .....	31
Maryland marble mill, plan of .....	98
Maryland quarries, methods in .....	79
Massachusetts marble, translucence of .....	13
Massachusetts quarries, methods in .....	88, 90
Matthew, G. F., on thrust faults.....	127
Merrill, G. P., on color of Colorado marble...	12
on deformation of rocks .....	17
on tests at Watertown Arsenal.....	131
on weathering of marble .....	30
Mica in marble .....	6
Mills. <i>See</i> Dressing plants.	
Milne, J., on earthquakes .....	131
"Monoclinic" bloc's. <i>See</i> Blocks.	
Monson, Mass., quarry, expansion of gneiss in	124
Monuments, marble for .....	20, 21
Mount Tabor, Vt., dolomite, color of .....	12
Mud pockets, formation of .....	20
N.	
New York City, abrasion of tombstones in ..	8
Niles, W. H., on expansion of gneiss .....	124
on stresses and strains .....	124-125
North Carolina quarries, methods in .....	80
Norway marble, color of .....	12, 35
O.	
Ontario marble, color of .....	12
mineral impurities in .....	6

	Page.
Onyx, origin of.....	7
use of.....	21
Overburden, determination of.....	39-40
limitation of quarrying by.....	49
removal of, cost of.....	39-40, 48, 49
methods of.....	46-48
P.	
Parian marble, translucence of.....	13
Parks, W. A., on color of Ontario marble.....	12
on determination of pore space.....	15
on durability of stones.....	19
on elasticity.....	17
on minerals formed by metamorphism.....	6
on permeability of marble.....	38
"Penetrating pulley," description of.....	71-72
use of.....	72
Pennsylvania, marble mill, heating of.....	95
Pentelicon marble, translucence of.....	13
Phosphorus in marble.....	6
Planers, use of.....	104
"Pocahontas" marble, color of.....	11
Porosity, determination of.....	14-16
effect of.....	38
Potash in marble.....	6
Power plants, compressed air.....	52, 53
hydroelectric, form for costs.....	152
operation of.....	50, 51, 52, 96
<i>See also</i> Colorado, Vermont.	
steam, operation of.....	52-53
Power transmission, methods of.....	51, 52-53
Proctor, Vt., marble near, composition of.....	5
Prospecting, cost of.....	45
geologic maps for.....	39
necessity for.....	39, 109
surface observations in, value of.....	40
with diamond drill.....	40
Pulley, penetrating. <i>See</i> "Penetrating pulley."	
Pyrite in marble, composition of.....	30
decomposition of.....	30, 31, 32
effect of.....	6, 30-31, 33, 118
hardness of.....	31
Pyrrhotite in marble, metamorphism of.....	31
occurrence of.....	30
Q.	
Quarries, drainage of.....	88
Quarry walls, arrangement of.....	109-110
Quarrying, cost of.....	108
economical operations in.....	70, 88
effect of uniform deposits in.....	49
efficiency in.....	88-89
improved methods of.....	65, 143
open-pit.....	84
safety devices.....	89
success in, factors of.....	43-49
undercutting in.....	87-88
waste in.....	108-122, 123, 136
result of strains.....	136
<i>See also</i> Channeling methods, Cost, Tunneling.	
Quartz in marble.....	6
R.	
Reade, T. M., discussion of strain phenomena.....	127
Records, block, use of.....	158-159
<i>See also</i> Cost keeping.	
Rensselaer, N. Y., rock faults at.....	128

	Page.		Page.
Renwick, W. G., on hole drilling.....	71	Tennessee quarries, methods in....	55, 68, 91, 139-143
on "penetrating" pulley.....	71-72	strain in.....	135
on statuary marble.....	20	Terrazzo, production of.....	122
Rift, definition of.....	14	Texture of marble. <i>See</i> Marble.	
"Right-angled" blocks. <i>See</i> Blocks.		Tickets. <i>See</i> Cost keeping.	
Ripley township, N. Y., anticlines in.....	127	Titanium, oxide of, in marble.....	6
Riprap, use of.....	120	Tombstones, abrasion of.....	8, 19
Road metal, crushed marble for.....	122	Tourmaline in marble.....	6
Roebing clips for guys.....	90	Transfer car, use of.....	100-101
Rubbing bed, driving of.....	103	Transportation of marble, methods of.....	87, 93-94
requisites for.....	103	Tremolite in marble.....	6
Rubble, marble used for.....	122	"Triclinic" blocks. <i>See</i> Blocks.	
Rudzki, M. P., on diminishing of elastic force.....	134	Troy, N. Y., rock faults near.....	128
on effect of temperature on plasticity....	134	Tuckahoe, N. Y., limestone from, composi- tion of.....	5
on elastic resistance of rock.....	135-136	Tunnel, application of term.....	84
S.		Tunneling, methods of.....	84-86, 87-88
Safety devices in quarries.....	89	waste in.....	110-111
Sand pumps, use of.....	99	Tunnels, lighting and ventilation of.....	87
Saw beds, haulage to.....	99-100	transportation of material in.....	87
removal of stock from.....	101-102	U.	
Saw gangs, construction of.....	98	Unsoundness in marble. <i>See</i> Channeling methods, Verd antique.	
Sawing, methods of.....	99	V.	
Saws, diamond, use of.....	104, 105, 106	Van Hise, C. R., on occurrence of chert.....	33
wire, mechanism of.....	71, 73	on weathering of fine-grained rocks.....	19
uses of.....	72-73, 74, 102	Van Horn, F. R., on anticlines in shales.....	128
Scabbling, methods of.....	92	on expansion of rock.....	131
Serpentine marble. <i>See</i> Verd antique.		Verd antique, origin of.....	7
Shops. <i>See</i> Dressing plants.		unsoundness in.....	29
Signals for hoisting.....	92	uses of.....	21
Silica in marble.....	6, 33-35	Vermont, granite, compression in.....	128
hardness of.....	34	hydroelectric power plant in.....	51
source of waste.....	117-118	marble, color of.....	11
Silicates in marble, origin of.....	34-35	polishing of.....	104
Sioux quartzite, hardness of.....	8	texture of.....	13-14
wind polish of.....	8	translucence of.....	13
Slick seams. <i>See</i> Channeling methods.		marble mill, heating system of.....	95
Sodium, oxide of, in marble.....	6	methods in.....	101
Solubility of marble.....	11	plan of.....	97-98
Specific gravity, determination of.....	9-10	water power for.....	96
Statuary, marble for.....	20, 21	power plants in.....	51
Stokes, H. N., on metamorphism of pyrite and marcasite.....	32	quarries, methods in.....	55, 57, 58, 60, 81, 82, 90
Stonington, Me., compression in quarries at.....	130	Vinal Haven, Me., strain breaks in granite at.....	130
Strahan, Aubrey, on "explosive slicken- sides".....	126	Vogt, J. H. L., on color of Norwegian marble.....	12, 35
Strain, behavior of rock under.....	133-135	W.	
cause of waste.....	123	Waldo County, Me., compression in quarries in.....	128
causes of.....	130, 136	Waste in marble, causes of.....	111-112, 117-119, 136
effect of quarrying on.....	135-136	utilization of.....	119-122
relief from.....	135, 136, 137, 139-143	<i>See also</i> Channeling methods; quarrying.	
Strain phenomena, examples of.....	129-130	Waterford, Conn., compression in quarry at.....	125
references in literature to.....	124-129	Weathering. <i>See</i> Marble, disintegration.	
Strength of building stones.....	17	Wedging, cost of.....	79
Stripping. <i>See</i> Overburden.		influence of rift on.....	79-80
Sulphur in marble.....	6	methods used in.....	65, 76, 77-79, 81, 82, 114
Superintendent, qualifications of.....	146	West Rutland, Vt., quarries, methods in.....	86
Swanton, Vt., marble, hardness of.....	8	Wind polish on marble.....	8
T.		Wollastonite in marble.....	6
Tennessee, "boulder quarries" in.....	69	Woodbury Center, Vt., quarry, strain in.....	130
Tennessee marble, color of.....	11-12	Woodworth, J. B., on rock faults.....	128
erosion of.....	20		
Tennessee marble plant, motor power for.....	96		







DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

PROSPECTING AND MINING OF COPPER ORE  
AT SANTA RITA, NEW MEXICO

BY

DONALD F. MACDONALD

*Geologist, United States Geological Survey*

AND

CHARLES ENZIAN

*Mining Engineer, Bureau of Mines*

---

Cooperation with the United States Geological Survey



WASHINGTON  
GOVERNMENT PRINTING OFFICE

1916

The Bureau of Mines, in carrying out one of the provisions of its organic act—to disseminate information concerning investigations made—prints a limited free edition of each of its publications.

When this edition is exhausted copies may be obtained at cost price only through the Superintendent of Documents, Government Printing Office, Washington, D. C.

The Superintendent of Documents *is not an official of the Bureau of Mines*. His is an entirely separate office and he should be addressed:

SUPERINTENDENT OF DOCUMENTS,

*Government Printing Office,*

*Washington, D. C.*

The general law under which publications are distributed prohibits the giving of more than one copy of a publication to one person. The price of this publication is 25 cents.

*First edition. September, 1916.*



# CONTENTS.

---

	Page.
Introduction.....	1
Acknowledgments.....	2
Location and general geography of Santa Rita district.....	2
Geologic conditions governing mining in Santa Rita district.....	3
Ore deposits.....	4
Adaptation of mining methods to geology and topography.....	5
History of mining in Santa Rita district.....	7
Present conditions and extent of property.....	9
Prospecting and locating ore bodies.....	9
Churn drilling and sampling.....	10
Type of drill used.....	10
Methods of drilling.....	10
Errors in churn-drill sampling.....	10
Operation and efficiency of prospect drilling.....	11
Cost of prospect drilling.....	14
Assay records and ore classification.....	14
Cross-sectioning the ore bodies.....	16
General considerations in delimiting ore bodies.....	18
Methods of estimating ore.....	18
Assay plan.....	19
Preliminary estimates.....	19
Estimating ore by triangular-prism method.....	20
Formulas for estimating ore.....	28
Formula for determining lowest grade of rock that may be classed as ore.....	28
Formula to determine whether ore should be smelted in the crude or milled and the concentrates smelted.....	30
Sampling and assaying methods.....	35
Prospect-hole samples.....	35
Gold assays.....	36
Blast-hole and pit samples.....	36
Steam-shovel samples.....	36
Sample shop and assay shop.....	37
Number and cost of assays.....	37
Forms used in assay shop.....	38
Method of sampling old dumps.....	40
Details of mining in Santa Rita district.....	40
General organization of operating company.....	40
Mining and estimation of ore by benches.....	41
Blast-hole drilling.....	42
Types of churn drills used.....	42
Operation and efficiency of blast-hole churn drills.....	43
Costs of blast-hole churn drilling.....	44
Types of tripod drills used.....	45
Costs of drilling blast holes by air drills.....	45
Use of hammer drills.....	45

## Details of mining in Santa Rita district—Continued.

	Page.
Blasting.....	46
Explosives used.....	46
Storage and handling of explosives.....	47
"Springing" or chambering holes.....	48
Charging and firing.....	51
Factors affecting blasting efficiency.....	55
Secondary or "dobe" blasting.....	57
Blasting costs.....	57
Blasting accidents.....	58
Safety rules.....	58
"Gopher" blasting.....	59
Loading ore.....	63
Transportation of ore.....	64
Crushing of coarse ore.....	66
Mine drainage.....	68
Effects of slopes of open cut.....	68
Compressor plant.....	70
Central power plant and power distribution.....	70
Santa Rita power and shop equipment.....	73
Power costs.....	74
Water supply at Santa Rita.....	75
Water-softening plant.....	76
Fuel.....	76
Mining costs.....	78
Milling.....	78
Hurley water supply.....	84
Milling costs.....	85
Repair shops at Hurley.....	86
General engineering details.....	87
Monthly estimates and statements.....	87
Monthly, quarterly, and annual statements by engineering department...	91
Health and sanitary measures.....	94
Hospital at Santa Rita.....	94
Accidents reports.....	95
Sanitary equipment.....	97
Rescue and first-aid work.....	97
Recreation.....	98
Time keeping, accounting, and warehouse methods.....	98
Time keeping.....	98
Accounting.....	105
Warehouse methods.....	111
Publications on mine accidents and methods of metal mining.....	116

## FORMS.

---

FORM 1. Form used by driller in reporting to mine office the work done by his crew.....	12
2. Form used in compiling monthly report from data in daily driller's reports, as shown in Form 1.....	13
3. Form used in assay record of drill-hole sample.....	15
4. Specimen cross-section assay record of results of sinking three prospect churn-drill holes to determine copper content of ore bodies..	16
5. Typical cross section of ore body.....	18
6. Form of general assay certificate issued by assayer.....	38
7. Form used by assayer to report on steam-shovel samples.....	39
8. Form for presenting comparative data regarding steam-shovel samples	39
9. Form used in making daily report on mine samples.....	40
10. Form filled in by applicant for position as skilled laborer.....	99
11. Form used in notifying timekeeper of the hiring of a new employee..	99
12. Register kept at time office.....	100
13a. Form used by time inspector for recording "time" of miscellaneous employees.....	100
13b. Form used by time inspector for recording "time" of steam-shovel crews.....	100
13c. Form used by time inspector for recording "time" of locomotive crews.....	101
14. Form used in recording "time" of men employed on special jobs, the expense of which is to be charged to certain accounts.....	102
15. Form used in connection with special jobs by skilled laborers, chargeable to certain accounts.....	102
16. Form used in connection with work done by a special gang and chargeable against each of several accounts.....	103
17. Form used in time office for keeping each man's time for a month...	103
18. Slip given by foreman to employee who is discharged or quits.....	103
19. Slip handed by timekeeper to employee who leaves during month..	104
20. Personal-record card.....	104
21. Form used in recording data on labor-distribution cards.....	105
22. Form used in recording total labor charges for each month, showing distribution.....	106
23. Operating-cost sheet.....	108
24. Form used in making statement of invoice.....	112
25. Warehouse-record sheet.....	112
26. Specimen sheet from price-and-invoice book.....	113
27. Form used in order on storekeeper for material.....	113
28. Form used in recording daily issues of supplies.....	113
29. Form used in summarizing daily records from Form 28.....	114
30. Form for keeping record of powder supply.....	115

## ILLUSTRATIONS.

	Page.
PLATE I. Map of part of New Mexico, showing region around Santa Rita.....	2
II. Contour map of region around Santa Rita before mining was commenced.....	4
III. Topographic map of ore deposits at Santa Rita, N. Mex.....	6
IV. Organization chart of the Chino Copper Co.....	42
V. <i>A</i> , View of a bench or terrace of ore, showing method of mining; <i>B</i> , Pile of coal containing wooden ventilating stacks to prevent spontaneous combustion.....	42
VI. <i>A</i> , Entrance to underground magazine; <i>B</i> , Exploding a springing charge in a bank hole.....	48
VII. <i>A</i> , Loading a toe hole; <i>B</i> , Face of shovel cut after firing blast holes, showing large boulders on which "dobe" charges are being laid..	54
VIII. <i>A</i> , Steel ore car; <i>B</i> , Steel ore car in dumping position.....	64
IX. <i>A</i> , Plant for crushing coarse ore, and loading bin, showing cars and loading tracks; <i>B</i> , View of part of the grizzly shown in Plate IX, <i>A</i> .	68
X. <i>A</i> , Face of excavation, showing jointing; <i>B</i> , Another view, showing jointing.....	70
FIGURE 1. Diagram showing character and distribution of reports made in con- nection with drill holes.....	11
2. Method of estimating classes of ore from drill records so as to show them on cross sections.....	17
3. Methods of dividing ore area into triangles.....	23
4. Side and end views of improvised furnace for evaporating drill sam- ples of ore.....	35
5. Plan of typical underground magazine in Santa Rita district.....	47
6. Longitudinal and transverse cross sections of "springing gage".....	49
7. View of chambered drill hole, showing how "springing gage" is used.	50
8. Arrangement of fuses for different primers.....	50
9. "Springing" charges in bank holes in dry soil or soft rock.....	51
10. Typical blasting charge in bank hole.....	53
11. Arrangement of detonator and electric wires for exploding charge..	54
12. Results of firing bank and toe holes in positions shown.....	55
13. Results of firing blast holes in firm rock.....	56
14. Placing of "gopher" holes for large blast.....	61
15. Horizontal curvature and lengths of different sizes of pipe of Pinder well pipe-line.....	75
16. Flow sheet of Santa Rita water-softening plant.....	77
17. Flow sheet of plant for crushing coarse material, Hurley mill.....	79
18. Flow sheet of a section of operations at Hurley mill.....	81
19. Plan map of excavation and of filling during one month.....	88
20. Plotted cross section of month's excavation.....	88



# PROSPECTING AND MINING OF COPPER ORE AT SANTA RITA, N. MEX.

---

By DONALD F. MACDONALD and CHARLES ENZIAN.

---

## INTRODUCTION.

It is a far cry from the small and uncertain mining efforts of 50 years ago to the splendidly equipped and solidly financed organizations that now handle thousands of tons of ore per day. The old mining was largely speculative; the new is practically on a manufacturing basis. In the old days of mining the aim was, chiefly, to find comparatively rich ore; modern practice looks more toward efficiency in mining and milling, so that vast tonnages of low-grade material may be worked at a profit. The old operators had faith and hoped for good returns; the new ascertain with a fair degree of precision what the available tonnage is, what the cost of mining and treating the ore will be, the time necessary to exhaust the deposit, the capital required, and the net profits to be made. The old mining was financed on a highly speculative basis, and the value of mining stock was largely controlled by the imaginations of the promoters and of the individual holders. In the new régime the financing of mines is often as conservative as the financing of manufactures or railroads, and the value of stock, quoted on the leading exchanges, is controlled by the carefully estimated profits in sight. Of course, with every legitimate industrial enterprise go certain business risks, and mining is not exempted from these. Then, too, valueless mining as well as valueless industrial stocks are sometimes offered for sale by the intentionally dishonest, or the ignorant. However, the prospective purchaser of a mine nowadays, if he is willing to pay for the services of reputable mining engineers, can obtain information on which to base an opinion as to the quality of his investment.

The relatively recent development of cheap and efficient methods of mining and of ore dressing has given value to low-grade ore bodies that a few years ago were almost valueless. The iron mines of northern Minnesota have been fruitful in developing efficiency in mining methods, notably the open-cut methods in which steam shovels strip off the overburden and then load the ore directly into railroad trains for transport to the furnace. In 1907 open-cut mining was applied to the low-grade copper ores of Bingham, Utah, with splendid results, and since then it has won a most important place in the mining of large masses of low-grade ores in several other mining regions, especially at Santa Rita, N. Mex.

Efficiency in mining as in other industries implies attention to detail, and the larger the scale of operations the more important may details become. For this reason the accompanying study of the methods used at Santa Rita will, it is believed, be of help to all engineers who have occasion to handle large masses of material.

#### ACKNOWLEDGMENTS.

Any merit that this report may have must be largely attributed to the splendid opportunity for study afforded the authors by the Chino Copper Co. To the manager, Mr. John M. Sully, the authors are deeply indebted for kindly interest and every facility to investigate the methods in use at the mines and mill of the company. For many courtesies and much help they are also under obligations to Horace Moses, superintendent; L. E. Foster, assistant superintendent; M. J. McGrath, chief engineer; Harry Thorne, explosives foreman, and many other officers of the Chino company's mine as well as to W. H. Janney, superintendent, and W. T. MacDonald, assistant superintendent, of the Hurley mill. Nearly two months were spent by the authors in field work at Santa Rita in the summer of 1914.

The outline of the geologic conditions that follows is largely from notes furnished by Spencer<sup>a</sup> and by Paige;<sup>b</sup> others<sup>c</sup> have also furnished valuable data bearing on the mining conditions of the region.

#### LOCATION AND GENERAL GEOGRAPHY OF SANTA RITA DISTRICT.

Santa Rita is on the mountain-bearing plateau of southwestern New Mexico. It is nearly 150 miles northwest of El Paso, Tex., and about 50 miles in the same general direction from Deming, N. Mex. It lies about a dozen miles east of Silver City, N. Mex., and is on Santa Rita Creek, not far from where the latter has its rise in the southeastern foothills of the Black Range. A branch of the Santa Fe Railroad connects Deming, N. Mex., on the Southern Pacific main line, with Silver City. From Whitewater, N. Mex., on this branch line, a subbranch leads to Santa Rita and to Fierro, N. Mex. (See maps, Pls. I and II.)

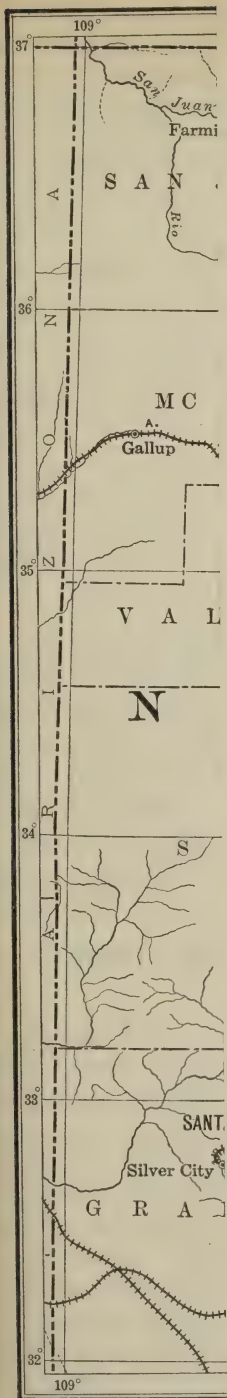
Santa Rita is in the central mining district of Grant County. In its vicinity the plateau is about 6,300 feet high, but some of the neighboring peaks reach altitudes of nearly 8,000 feet.

The climate is dry, bracing, and sunny. Even in the bright hot summer months the nights are cool. In winter, clear, cool weather, with occasional snow, adds to the stimulating effect of the climate.

<sup>a</sup> Spencer, A. C., A detailed report on the geology of Santa Rita, in course of publication by the United States Geological Survey.

<sup>b</sup> Paige, Sidney, Silver City folio (No. 199), Geol. Atlas U. S., U. S. Geol. Survey, 1916, 20 pp.

<sup>c</sup> Lindgren, W., Graton, L. C., and Gordon, C. H., The ore deposits of New Mexico: U. S. Geol. Survey Prof. Paper 68, 1910, 361 pp.







MAP OF PART OF NEW MEXICO, SHOWING REGION AROUND SANTA RITA.



The following table gives the maximum and minimum temperatures and monthly evaporation for the year 1914. The monthly mean of the daily maximum and minimum temperatures is given under "mean maximum" and "mean minimum."

*Data on temperature and precipitation at Santa Rita, N. Mex., during 1914, by months.*

Month.	Temperature. <sup>a</sup>				Evapo- ration.	Precipitation. <sup>b</sup>		
	Maxi- mum.	Mini- mum.	Mean maxi- mum.	Mean mini- mum.	During month.	Total.	During month.	Total.
1914.	° F.	° F.	° F.	° F.	Inches.	Inches.	Inches.	Inches.
January.....	64	14	52.27	28.15	4.329	4.329	0.58	0.58
February.....	62	22	51.39	28.95	4.266	8.595	0.43	1.01
March.....	70	24	57.63	32.66	7.448	16.043	0.66	1.67
April.....	78	30	66.88	41.18	11.507	27.550	0.02	1.69
May.....	84	31	74.23	48.27	12.416	39.966	1.00	2.69
June.....	91	48	81.82	57.50	12.253	52.219	1.51	4.20
July.....	85	52	78.11	58.76	7.612	59.831	7.51	11.71
August.....	88	53	80.98	58.66	9.151	68.982	2.98	14.69
September.....	83	46	76.77	54.12	8.214	77.196	0.75	15.44
October.....	78	37	65.27	43.03	5.964	83.160	3.58	19.02
November.....	64	30	57.73	36.48	3.584	86.744	0.93	19.95
December.....	50	15	42.94	27.15	1.636	88.380	4.70	24.65

<sup>a</sup>Elevation of gaging station, 6,311.5 feet.

<sup>b</sup>Snow reduced to equivalent rainfall.

The streams are not large, and the country is not particularly well watered, either for mining or for agriculture. Marketable timber is scarce except in the higher mountains, and even there it is inclined to be scrubby. The timber used at the Santa Rita mines is brought in by rail, as the available local supply, never great, was exhausted some years ago.

Santa Rita and the neighboring regions offer an attractive field to the prospector; the climate is delightful, the country open and easy to traverse, and the rock outcrops fairly favorable.

### GEOLOGIC CONDITIONS GOVERNING MINING IN SANTA RITA DISTRICT.<sup>a</sup>

The ore bodies at Santa Rita are associated with quartz-monzonite and quartz-monzonite porphyry intrusions, which cut limestones, sandstones and shales. These intrusives are of post-Cretaceous date, and are older than tuffs, rhyolites, and andesites in the vicinity, which are regarded as of Miocene age. Of the three types of intrusives near the ore, the oldest and the one constituting the largest mass is a quartz-porphyry; the next in point of age is an essentially even-grained quartz-bearing monzonite, containing idiomorphic feldspars and occurring as a stock of moderate size; the youngest is a porphyritic rock, locally characterized by large crystals of quartz, which occurs in the form of dikes. The sedimentary series cut by

<sup>a</sup> Notes on the geology of the region were furnished by Dr. A. C. Spencer, whose complete report on the geology of the Santa Rita district will be published by the United States Geological Survey.

these intrusives consists of limestones of Carboniferous (Pennsylvanian) age, and of Cretaceous shales and sandstones.

The alteration of the rocks in the vicinity of the ore deposits has been marked. Within a rudely elliptical area, measuring about 7,000 feet east and west and more than 10,000 feet north and south, the prevolcanic rocks were thoroughly permeated by metamorphosing solutions. These solutions have sericitized the shales and have charged them with pyrite; they have silicified and pyritized the sandstones, and have changed the limestones in many places to chlorite-sericite rocks, carrying large amounts of pyrite and locally much magnetite. All the igneous rocks are somewhat mineralized, as shown by the presence of pyrite. The older quartz-porphyry is practically everywhere completely sericitized, but this sort of alteration is much less general, and is locally lacking in the younger intrusives.

In the main, the mineralization is of the type that is produced by general metasomatic alteration of the rocks. Highly individualized veins are essentially lacking, though reticulating fractures in the rocks are locally healed by quartz fillings. The date of the mineralization is not precisely known, but on the south the altered rocks are overlapped by unmetamorphosed volcanics, indicating that the mineralization was essentially complete before the extravasation of the latter in Miocene time.

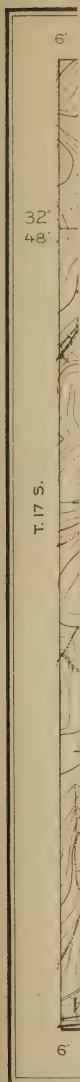
#### ORE DEPOSITS.

The copper ores of Santa Rita are mainly low-grade deposits in which the copper minerals are disseminated through the rocks, and various rocks, both sedimentary and igneous, are contained in the ore bodies. Metamorphism and primary sulphide deposition resulted from the action of hot solutions of magmatic origin. Almost everywhere the altered rocks carry at least small amounts of copper, but material of ore grade has been formed mainly through the enrichment of relatively lean rock by the downward movement of water.

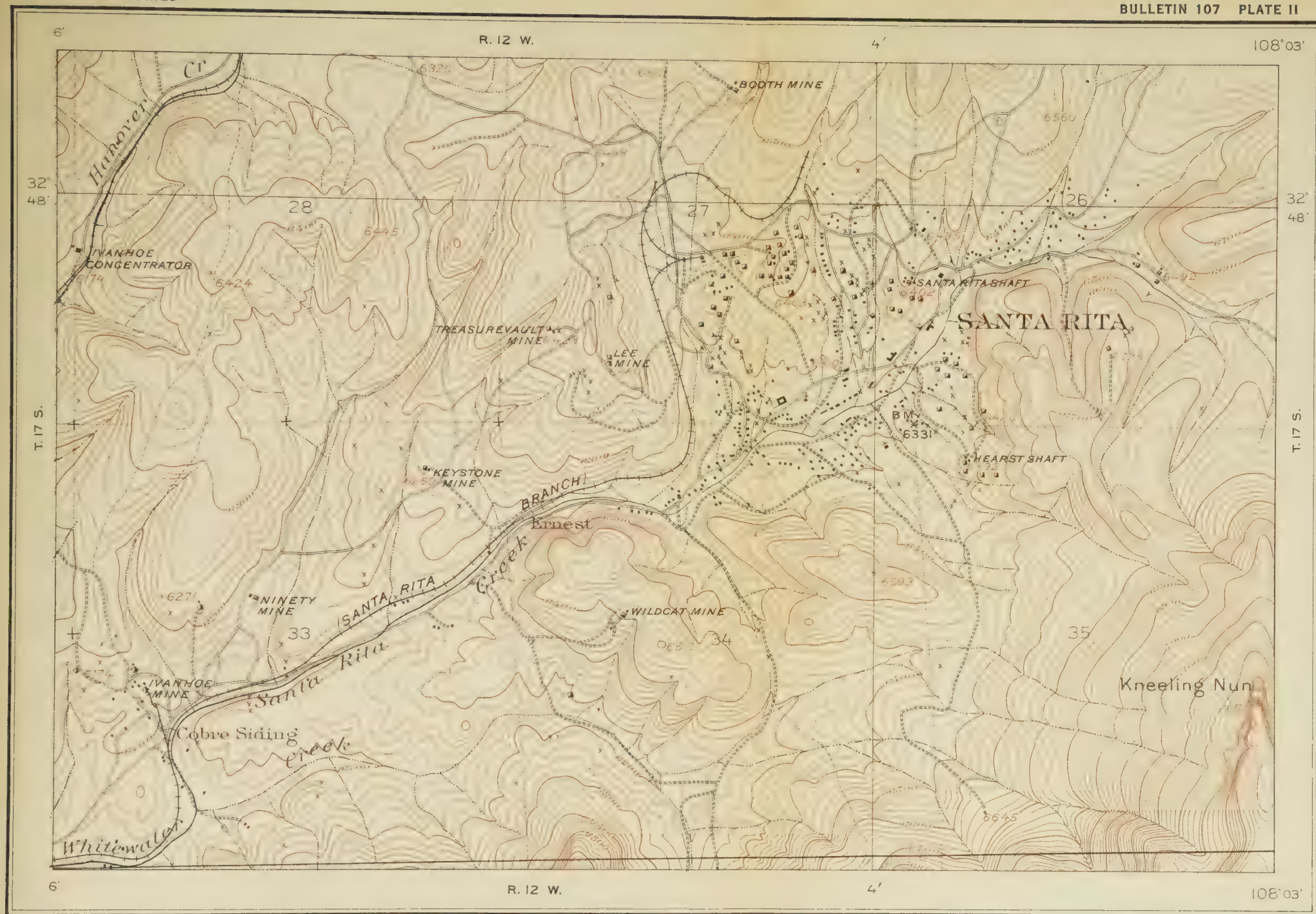
The principal copper mineral of the sulphide ores is chalcocite, but metallic copper and cuprite are locally present. Chalcopyrite, though seemingly not abundant in the ores, must have been the primary mineral from which most, if not all, of the secondarily deposited copper has been derived. Malachite occurs in much of the weathered capping, and is intermixed in varying proportions with other copper minerals in the upper parts of the sulphide ore bodies.

Rich copper ores that were mined in former years came from a series of ore bodies disposed along a curving belt 400 to 800 feet wide and about 6,000 feet long. This ore belt is comprised in the wider and more extensive zone which includes the principal ore bodies of much lower average grade that have been developed by the Chino

BURI

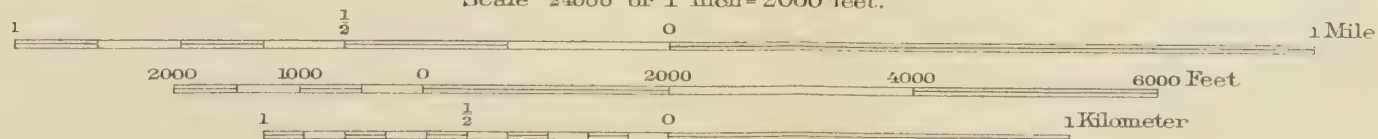






# CONTOUR MAP OF REGION AROUND SANTA RITA BEFORE MINING WAS COMMENCED

Scale  $\frac{1}{24000}$  or 1 inch = 2000 feet.



Contour interval 20 feet.

Datum is mean sea level.

TRUE NORTH  
MAGNETIC NORTH  
APPROXIMATE MEAN  
DECLINATION 1907.

Surveyed in 1907  
by the U. S. Geological Survey



Copper Co. As now outlined (see Pl. III) the ore zone is an approximately elliptical annulus 500 to 1,000 feet wide, with an inside shorter diameter of 2,000 feet and a longer diameter of 3,000 feet. The principal axis of the ellipse trends northwest. The ores are of variable composition, as the different rocks mentioned have entered into the ore bodies, and each rock has been differently affected by the metamorphosing agents.

In the main the results of churn-drill prospecting show that the ore bodies are rather definitely though not sharply limited downward, and this, together with the fact that chalcocite replaces pyrite, proves that the ores owe their present copper content partly to processes of enrichment due to the downward movement of surface waters.

The weathered overburden that caps the sulphide ores varies in thickness from a few feet to 150 feet or more, the average being about 85 feet. In general the ore bodies measure more than 100 feet vertically, but what may be called the bottom surface is extremely irregular. Thus in one section of the ore area, material carrying chalcocite and metallic copper has been found continuing to depths as great as 1,300 feet, and in several places the ore bodies are 400 to 500 feet thick.

The variations in thickness, and the fact that the segregations that are large enough and of sufficiently high grade to constitute ore occur in a rather definite zone, are thought to be due to differences in the conditions of underground drainage.

The variations in the mineral constitution of the ores are wide. Different sorts and grades of ore are rather intricately intermixed, so that maintenance of the output of a mine at any average mineral composition is always difficult and at times impossible. The chief variations of practical moment are, first, in the relative amounts of copper minerals to pyrite and magnetite, and, second, in the proportions of heavy copper minerals to malachite. In the milling of the ores a high iron content—that is, a high proportion of pyrite and magnetite—tends to a low ratio of concentration, and any large proportion of malachite results in a marked decrease in the percentage of copper recovered.

#### ADAPTATION OF MINING METHODS TO GEOLOGY AND TOPOGRAPHY.

The methods of prospecting and mining adopted in the Santa Rita district have been determined by the geologic features of the ore deposits, and the general scheme of excavation, ore delivery, and waste disposal has been nicely adjusted to the topographic situation.

By reason of the wide extent of the ore bodies as compared with their depth, they were well suited to churn-drill prospecting for the

determination of ore tonnages and grades of ore. The altered and generally soft nature of the rocks resulted in low drilling costs, and the relatively even grade of the ore favored accuracy in determining the copper content of the ores.

The feasibility of open-cut mining by means of steam shovels depends on the size of the ore bodies and the thickness of the overlying waste.

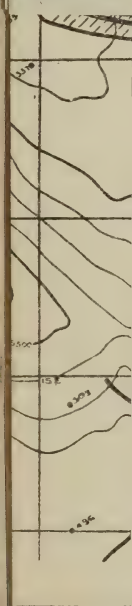
Because the altered rocks that contain the ore and constitute the gangue are soft, drilling is easy. These rocks are much jointed and break readily when blasted.

Any appreciable addition to the amount of waste material that must be moved, owing to large slides, is not anticipated. In general, the overburden of soil, or unconsolidated *débris*, is shallow, and everywhere such material is essentially free from percolating water. Overburden consisting of weathered rock is also well drained, and both materials stand well without "slaking" in any important degree. Although the ore and the weathered rock contain abundant sericite, and when finely broken develop much colloidal material, in place they are generally strong enough to stand with fairly steep slopes for all depths that are reached by open pits.

The mines are situated in a basin which is a broadened section of the valley of Santa Rita Creek.

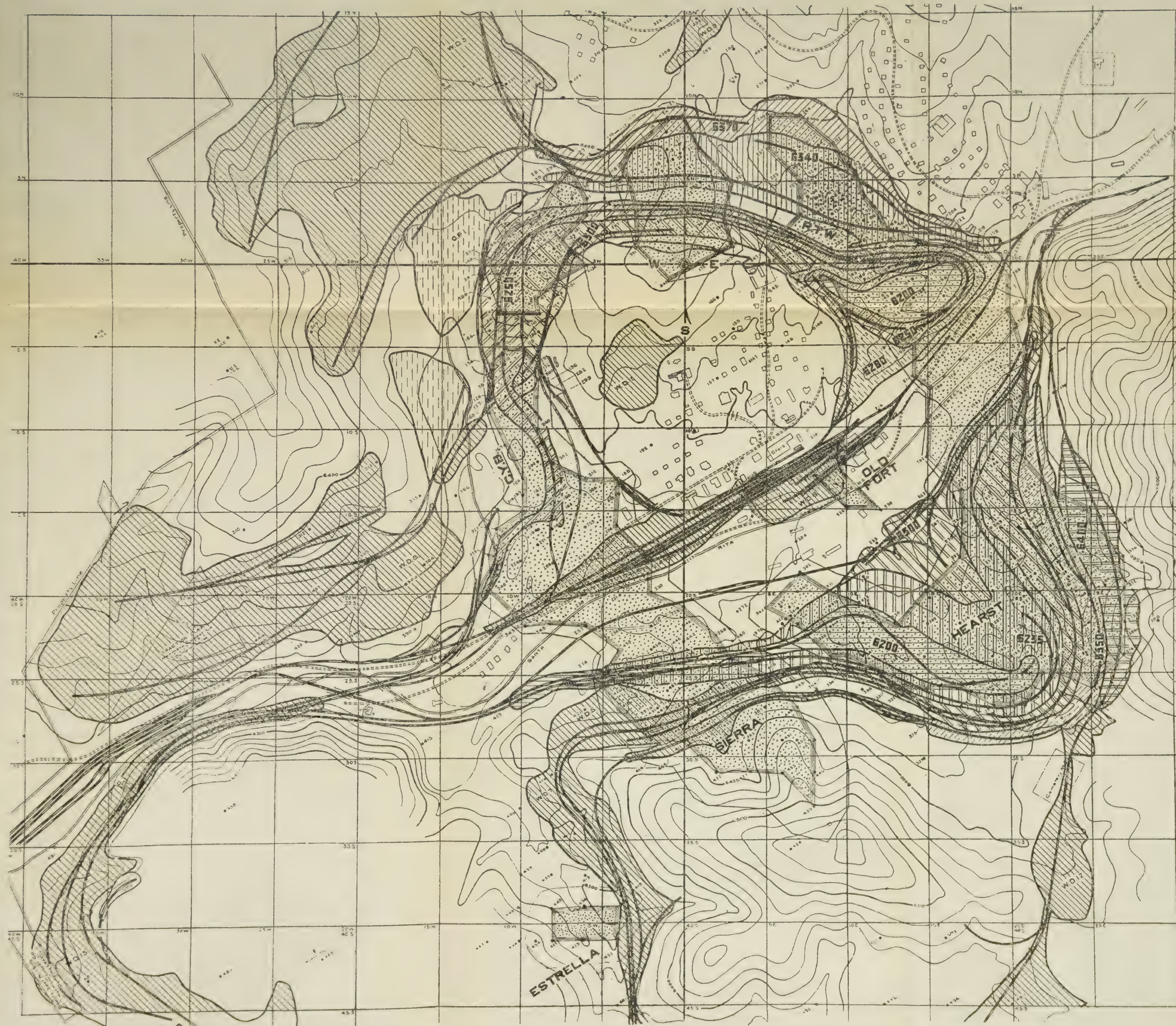
In plan the outline of the Santa Rita mine workings is like that of a jew's-harp, two great cuts forming the flaring loop, the block left to carry the creek forming the tongue (see Pl. III). The creek flows southwest, leaving the ore-bearing area 4,000 feet beyond the point of entry and at an elevation 120 feet lower. The inclined approaches that permit locomotive haulage in removing the ore from the deep pits have grades opposed to the general surface slope. These approaches diverge at a point more than 1,200 feet down stream from the westerly edge of the ore area. The length of haul from the far end of each of the mine pits to the common track that serves the crushing plant is thus greater than 1 mile, so that considerable depths can be directly attained. Eventually spirals or switchbacks will be required and possibly the use of hoisting planes or Shay geared locomotives may be advantageous. At present the development plans of the company do not contemplate that any mining will be done by means of steam shovels below an elevation of 6,050 feet, which is approximately 150 feet below the points where the pit approaches intersect the ground surface.

The crusher is situated near the mouth of the canyon-like valley which drains the Santa Rita basin, the delivery track being at a level about 80 feet above the assembly yard. From this yard the ore trains go to the concentrating plant at Hurley by way of the Atchison, Topeka & Santa Fe Railroad.



and the track

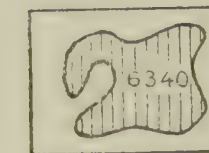
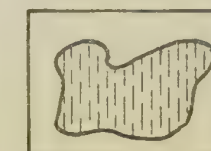




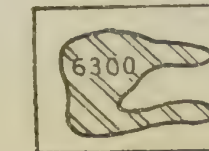
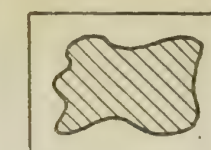
## LEGEND



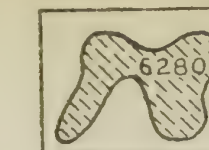
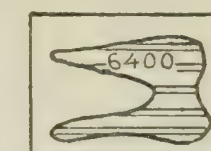
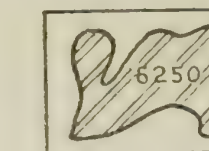
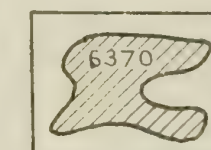
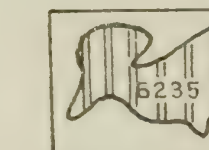
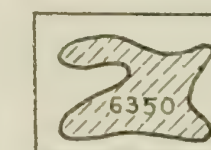
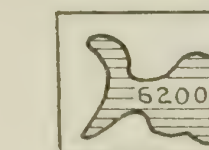
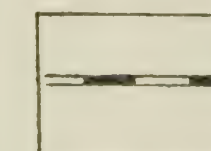
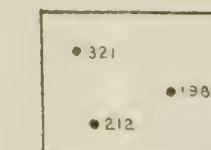
Ore bodies

Bench cut to  
elevation 6340

Ore-pile areas

Bench cut to  
elevation 6300

Dumps

Bench cut to  
elevation 6280Bench cut to  
elevation 6400Bench cut to  
elevation 6250Bench cut to  
elevation 6370Bench cut to  
elevation 6235Bench cut to  
elevation 6350Bench cut to  
elevation 6200Railroad tracks  
completedRailroad tracks  
projectedProspect drill  
holesBench elevations  
referred to  
sea level

TOPOGRAPHIC MAP OF ORE DEPOSITS AT SANTA RITA, N. MEX.

Shows relations of prospect drill holes to coordinates, placed every 200 feet, and to ore bodies, which are outlined; the methods of working the ore bodies by benches and the trackage on each bench are indicated.



Whereas the ore tracks necessarily converge for the purpose of delivery, the waste tracks are in the main divergent. The principal dumping ground is in the valley of Whitewater Creek, south of the mines, but most of the waste from the upper levels of the northern section has been deposited within the Santa Rita basin west and northwest of the mines.

Because of the semiarid character of the climate, current ground-water seepage is slight in comparison with the extent of the workings and pumping costs are correspondingly low. At times of heavy rains the lowest levels in the pits become flooded, but little time is lost from this cause. Continued rains, which are seldom experienced, result in reduced operating efficiency, mainly by causing track troubles.

For some time to come a channel for the creek will be maintained directly across the Santa Rita basin, but eventually, in order to remove large amounts of ore, the diversion of the stream will be necessary. Occasionally after heavy rains the volume of water carried by the creek is large, but the mine pits have been so planned that they are in no danger of being flooded from this source.

The Chino company has, for convenience of reference, grouped its property into six ore bodies, as follows: C. Y. B., Sierra, Hearst, R. W. T., Old Fort, and Estrella. This grouping, however, is somewhat artificial and the divisions do not correspond in every case with actual breaks in the mineralized area. In fact some of the ore bodies are somewhat merged with each other and some are separated by areas of mineralized but low-grade rock. Their annular grouping around a relatively barren core, their extent, and their general relations are shown by Plate III.

### HISTORY OF MINING IN SANTA RITA DISTRICT.<sup>a</sup>

It is said that copper was first mined at Santa Rita by the Aztecs. It was probably one of the places where they procured the native metal. About 1800 a friendly Apache chief showed the croppings near the present Romero shaft to Col. José Manuel Carasco, the officer in charge of the Spanish military posts in New Mexico. Col. Carasco interested Don Manuel Francisco Elguea, of Chihuahua, a wealthy merchant banker and subdelegate to the Spanish Court, and the latter obtained a concession to the property under Mexican law. Four years later Elguea purchased Carasco's interests and began to open up the property. The copper produced found a ready market with the Mexican Government for coinage. Almost until the early nineties the region was infested with warlike Apache Indians, who were not subjugated until almost exterminated; hence the Spaniards

<sup>a</sup> The chief source of information regarding past conditions is a manuscript report by Mr. John M. Sully, manager of the Chino Copper Co.

had to build a fort and maintain a military force to protect the camp. The fort was a triangular structure with heavy built adobe towers guarding each of its apices, and these were joined by strong adobe walls. One of the old towers is still standing and until recently was used as a jail. Other remnants of these ancient fortifications are still extant.

Rough smelting works were built and from the sulphide and oxide ores mined copper metal was extracted in a crude way. This was carried on mule trains, under military convoy, to Chihuahua and on to Mexico City. It is said that Elguea left at his death, in 1809, a considerable fortune. Later, his widow leased the property to Juan Onis. After a few years it was leased to Siqueiros, and was worked intermittently, owing to Indian outbreaks, until the late fifties, when Indian hostility finally forced its abandonment.

Gen. Sibley, in command of Confederate forces from Texas, held this region for a short time. Efforts, mostly unsuccessful, were then made to work the mines.

In 1873 the dawn of a more prosperous period was at hand, for M. B. Hayes, connected with the first smelting works in Colorado, began work at the present Romero mine. A 248-foot shaft was sunk, and a furnace, which proved inadequate, was erected. A shipment of 40 tons of picked ore and imperfectly smelted copper was hauled by teams 700 or 800 miles to a railroad station in Colorado and forwarded to the Baltimore Copper Works and to the Revere Copper Works, at Point Shirley, Mass. In October, 1873, Mr. Hayes and associates obtained title to, and possession of, the property from the sole surviving heirs of the Elguea family, who were settled through Mexico and Europe. Until the heirs had transferred their interests the United States Land Office had refused to issue patent to the property, because the Elguea family had rights under the treaty between the United States and Mexico. Later, patent was taken out by the Hayes interests, so as to make the title more secure and to guard against future litigation.

In 1881 the property was sold to J. Park Whitney, and a stamp mill, later burned, was erected near the foot of Romero Hill. In 1882 diamond drilling was done. Although the records of some of these holes are still extant, their exact location has not been ascertained.

The year 1891 saw the railroad completed to Hanover, a few miles away, and in 1899 a branch into Santa Rita was finished.

In 1897 Mr. Whitney gave a lease and bond to the Hearst estate, but in 1899 sold the property to the Santa Rita Mining Co., before the lease had expired. The new company developed some large bodies of high-grade sulphides and considerable native copper. After extraction of these, active operations were carried on, mostly by lessees, on a basis of 35 per cent to the owners when copper was 15

cents or more per pound, and 25 per cent when it was less than 15 cents. The Santa Rita company, however, continued to operate the mill, treating ore purchased from the lessees and low-grade ore from the dumps, formerly thrown out as waste. It is said that this old mill, which had a capacity of only 120 tons per day, left copper in the tailings to the amount of 4 per cent of the rock treated. It is therefore a far cry from the old methods to present-day milling efficiency. Lessees continued work until about 1906. Ore containing 10 per cent copper was sold by them, but ore with a lower copper content they threw on the waste dumps.

A new era dawned on this ancient mining district in 1906, for in that year systematic sampling of the property was begun by Mr. John M. Sully, the present manager of the Chino company. The old dumps and the old workings were sampled, and churn-drill sampling was begun.

#### **PRESENT CONDITIONS AND EXTENT OF PROPERTY.**

After an extensive report by Mr. Sully concerning the results of the sampling mentioned, the present company acquired the property, which now consists of 147 mining claims, including fractional and full claims, comprising 2,645 acres; 131 of these, aggregating 2,412 acres, are patented. In addition, the company owns 160 acres of agricultural land adjacent to its mining claims. The property contained 19 different shafts, aggregating over 4,000 feet of sinking, and these were connected by several thousand feet of underground workings.

This, then, was the beginning of the development as a result of which an old and inefficient mill that treated 120 tons of ore per day has been replaced by a modern mill, with a capacity of about 7,000 tons per day, which successfully treats ore containing less than 0.8 per cent of copper.

Stripping operations were started by the Chino company in October, 1910, the ore taken out being sent to stock piles. During August, 1911, the first shipment of ore was made to the mill. The total shipments during that month were nearly 14,000 wet tons.

#### **PROSPECTING AND LOCATING ORE BODIES.**

Preliminary to the systematic prospecting and delimiting of the ore bodies with churn drills, an accurate topographic map of the region, on a scale of 100 feet to the inch and showing a contour interval of 5 feet, was made. This map was then laid off into north-south and east-west coordinates at intervals of 100 feet, and a datum point at a coordinate intersection was established, for the purpose of locating, by coordinates referred to this point, all the drill holes that were to be put down, as for instance: Hole 10, north 650 feet and west 355 feet.



## CHURN DRILLING AND SAMPLING.

### TYPE OF DRILL USED.

For prospecting the ore body, No. 6 deep-hole Cyclone churn drills were used. They were fitted with steam power and were of the walking-beam type, using 4-inch to 8-inch bits, depending on the depth of hole and other conditions. In sampling low-grade deposits of relatively even tenor over a large superficial area, where the rocks are not hard, these drills give much better satisfaction than do diamond drills. Of course, churn drills can sink only holes that are vertical or approximately so, whereas diamond drills can be used to bore at any angle.

### METHODS OF DRILLING.

Drilling was begun as near to coordinate intersections as was convenient. Each drill was operated day and night by two crews, each working 12 hours, and consisting of a driller, his helper, and a sampler.

In the process of drilling, water was poured into the hole to form a sludge with the cuttings. As the drill was removed any mud adhering to it was washed back into the drill hole. In relatively barren surface material, trial samples were taken at intervals of about 5 feet, but below the surface material samples were taken every 3 feet, the distance being measured by marks on the drilling cable. When ready for sampling, a bailer with a valve in the bottom was lowered, and all the sludge cut in the sampling interval, 3 or 5 feet of drilling, was carefully bailed out. The sludge from the bailer was discharged into a launder, and from there was cut on what was in effect a Jones sampler. The material saved, about one-quarter of the total, was caught in a tub and, if too bulky, was again poured through the sampler; the bailer, launder, and tub were washed after use, and the washings were also run through the sampler. The final quartered sample, amounting to about a gallon, including the washings, was sent to an evaporating furnace, and, when dry, to the assay shop. The method of treatment there is described later. A part of each sample was caught and carefully panned, so as to indicate the kind and character of the rock and the minerals contained, and all the results were set down in the record or log of the hole.

### ERRORS IN CHURN-DRILL SAMPLING.

The errors peculiar to churn-drill sampling are as follows: (a) Deviation of the hole from the vertical; (b) breaking off at the point of drilling of a surplus of rich brittle mineral, thus giving a sample that is too rich; (c) concentration of heavy minerals in the bottom of the hole, so that they are not reached by the bailer, or are recovered



from the hole when it has reached a much lower depth where the rock may actually be barren; (d) caving of the rock from the sides of the hole.

Little trouble resulted from the deflection of holes, except in a few instances where the drills penetrated narrow zones of crushed rock contained between relatively hard, steep-sloping walls. The tendency was then for the drill to be deflected along the slope into the softer material. However, careful drilling usually remedied this tendency. A few deflections of as much as 4 or 5 feet per 100 occurred.

Concentration of heavy minerals in the bottom of a hole was not particularly troublesome, and was largely obviated by careful cleaning out of the bottom of the bore each time a sample was taken.

Caving was the most obvious cause of error, but was largely remedied by casing the hole and continuing it with a smaller bit. As a result of this procedure some of the holes that were 8 inches in diameter at the top were only 4 inches wide at the bottom.

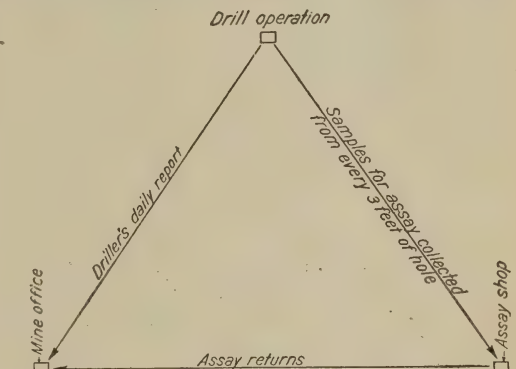


FIGURE 1.—Diagram showing character and distribution of reports made in connection with drill holes.

The accuracy of the churn-drill method of sampling was later tested by a few shafts and "raises" along some of the drill holes. The average results obtained from shaft samples and from drill-hole samples agreed closely. The churn-drill method of sampling other similar copper deposits in the Southwest has been proven comparatively accurate and satisfactory.

#### OPERATION AND EFFICIENCY OF PROSPECT DRILLING.

At the end of every shift, each driller submitted to the mine office an outline of what had been accomplished by his crew during the hours on duty. This was recorded on a special form, shown as Form 1 (p. 12).

In addition to the shift reports, each drill crew sent to the assay office samples quartered down from the total cuttings of every 3 feet of hole drilled. These were assayed and the returns were sent to the mine office as soon as completed. The different reports mentioned are more graphically shown by figure 1.

FORM 1.—Form used by driller in reporting to mine office the work done by his crew.<sup>a</sup>

## DAILY CHURN-DRILL REPORT.

## SAMPLES.

Depth.	Character of rock.		Minerals.	Sampling appliance.		Caving.			Is sample reliable? If not, state reasons under "General remarks."	True depth.	Length of sample.
Number of sample.	1. Quartz-diorite. 2. Andesite-porphyr. 3. Quartzite. 4. Limes or other rock. Leached or unleached. Altered or unaltered. Hard or soft. 5. Silicified porphyry. 6. Central city intrusive.	Color of sludge: Red or dark brown. Light yellow. Pink or purple. Light brown. Gray or white. Green or blue.	A Chalcocite. B Pyrite. C Metallic copper. D Oxides of copper. E Carbonates. F Chalcopyrite. G Magnetite. H Hematite. I Mica.	Split divider.	Tub.	No.	Yes, slightly.	Yes, considerably.			

## GENERAL REMARKS.

.....

.....

.....

WATER.—Depth reached..... Stands at depth of.....

.....

.....

.....

FAULTS AND CLAY SEAMS.—Depth below surface...ft. Width of fault material...ft.

SIZE OF BIT..... CASING: Size.....; depth.....

## EMPLOYMENT OF TIME.

Actual drilling hours..... Moving and setting up hours.....

Repair hours..... Casing or removing casing.....

Other delays.....

Driller..... Shift.....

Helper..... Date.....

Sampler..... Hole No. .... Drill No. ....

NOTE.—Enter time after each name.

<sup>a</sup> Size of form as printed by operating company, 8 $\frac{3}{4}$  by 13 $\frac{3}{4}$  inches; held in binder through two holes in top.

In the office the data from the daily drillers' reports were compiled into a monthly report (Form 2). The average performance and

FORM 2.—*Form a used in compiling monthly report from data in daily drillers' reports, as shown in Form 1.*

Date.	Shift.	Hole.	From—	To—	Number of feet drilled.	Hours drilling.	Hours moving.	Hours casing.	Hours reaming.	Hours repairing.	Other delays.	Total time.	Drill No. ....	
													Driller.	Helper.
1														
2														
3														
4														
5														
6														
7														
8														
9														
10														
11														
12														
13														
14														
15														
16														
17														
18														
19														
20														
21														
22														
23														
24														
25														
26														
27														
28														
29														
30														
31														
Totals														

<sup>a</sup> Size of form used by operating company, 8½ by 14 inches; upper half only, showing space for recording data covering day shift, is shown; lower half provides for similar data covering night shift. Held in binder through four holes at left.

efficiency of the churn drills used in prospecting the Santa Rita ore bodies can best be shown by the table following. In this the performance of two Cyclone No. 6 deep-hole drills is averaged for a period of 10 months. The figures are fairly representative of what has been accomplished by prospecting drills on the somewhat softened and locally sheared granitic and other rocks of Santa Rita. A similar table, showing in a comparative way the performance of all the drills, was made out for each month.

*Churn-drill record.*

[March to December, 1912—10 months.]

## DAY SHIFT.

Drill No.	Feet drilled.	Total working hours.	Per cent time drilling.	Per cent time delayed.	Per cent time repairing.	Per cent time moving.	Average feet per hour of drilling.
6.....	3,221	3,727	59.3	18.2	3.3	19.2	1.46
7.....	3,365	3,776	70.7	10.6	2.5	16.2	1.26

## NIGHT SHIFT.

6.....	3,071	2,598	78.9	16.4	3.5	1.2	1.50
7.....	2,947	2,850	84.4	12.3	2.2	1.1	1.23

## TOTAL.

6.....	6,292	6,325	67.0	17.5	3.4	12.1	1.48
7.....	6,312	6,626	76.4	11.3	2.4	9.9	1.25

## COST OF PROSPECT DRILLING.

To the miner the cost of prospecting is of great importance; hence cost data regarding churn drilling these ore bodies are, through the courtesy of the Chino Copper Co., presented in some detail, as follows:

*Cost data on churn or prospect drilling at Santa Rita, N. Mex.*

Drill Nos.	Month.	Cost of—						
		Labor.	Supplies.	Teaming.	Admin- istration.	Total.	Per foot.	Per shift.
	1912.							
6, 7, and 12.	May.....	\$2,106.14	\$494.03	\$142.18	\$349.28	\$3,091.63	\$2.31	\$22.90
6 and 7.....	June.....	2,029.35	661.09	80.25	270.23	3,040.92	2.29	26.44
6 and 7.....	July.....	1,997.43	451.20	112.75	229.67	2,791.05	2.23	24.48
6 and 7.....	August.....	1,969.57	496.14	129.00	209.12	2,803.83	2.00	23.56
6 and 7.....	September.....	1,953.78	442.02	150.25	234.24	2,780.29	2.13	24.56
6 and 7.....	October.....	1,823.99	506.98	153.95	185.25	2,670.17	2.68	23.95
6 and 7.....	November.....	1,775.52	619.47	134.75	238.54	2,768.28	2.42	27.82

The above table is valuable as a general index of the cost of churn-drill prospecting. Of course, in other places local conditions might cause considerable variation from the figures given.

## ASSAY RECORDS AND ORE CLASSIFICATION.

The assay returns were compiled on loose leaves (Form 3), which, when filled out, were put into strong permanent binders and kept as volumes. In the columns of this form the location of the sample in the hole was recorded under depth and its value when assayed by the permanganate method was shown in the "Perm. Cu." column.



FORM 3.—Form used in assay record of drill-hole samples.<sup>a</sup>

DRILL HOLE NO. ....  
 Coordinates.....  
 Time drilling .....

Depth.	% Perm. Cu.	% Met. Cu.	% Cu. total.	% Elect. Cu.	Averages.	Rock.	Minerals.	Sludge.

The results of electrolytic check assays were recorded under the heading "Elect. Cu." In the column headed "Averages," the ore was divided into classes, as follows:

*Ore classification used by Chino Copper Co.*

Class.	Copper content, per cent.	Designation.
P.....	0.00 to 0.79....	Poor.
L 1.....	0.80 to 0.89....	Low No. 1.
L 2.....	0.90 to 0.99....	Low No. 2.
F.....	1.00 to 1.24....	Fair.
G.....	1.25 to 1.49....	Good.
E.....	1.50 to 1.99....	Excellent.
S.....	2.00 upward....	Superlative.

Any native copper contained was recovered by screening or panning the crushed sample; this was weighed and the weight recorded in the column headed "Met. Cu." The sum of the native copper content and the copper determined by the permanganate method was then set down in the column headed "Cu. total." Notes on the kind of rock passed through, the minerals encountered, and the color and character of the sludge were set down in the columns headed "Rock," "Minerals," and "Sludge." It will be seen that this form gave a comprehensive record of each hole, including its location by coordinates, time of drilling, depth, values for every assay interval of 3 feet, and averages. However, to more thoroughly visualize the mineralized conditions indicated in each hole, its assay record was plotted on cross-section paper somewhat as shown on Form 4. Two ink lines to represent each drill hole were drawn as shown. The sections as determined by the groups of assay results were indicated by horizontal ink lines, as shown, each small square on the paper representing a depth, or sampling interval, of 3 feet; the figures opposite showed the percentage of copper that the rock contained.

In grouping the ore into the foregoing classes, every sudden and considerable change in values, if the record showed a change in the character of the sludge, was noted as a possible class boundary. Rock carrying less than 0.8 per cent of copper was considered of too low grade to be classed as ore, as is explained later. However, where small masses of waste occurred with the ore, in such a way that separation from the latter could not profitably be carried out, it was figured in as a diluent of the ore reserves.

#### CROSS-SECTIONING THE ORE BODIES.

Following the assay-record sheets came the work of making cross sections of the ore body. These were compiled from the data shown on the assay records and the drill hole records and show both north-south and east-west cross sections of the ore bodies at intervals of 100 to 200 feet, depending on the spacing of the drill holes and the data that they furnished. Where the holes were several feet away from a cross section, the data from them were projected to the section and recorded there, with a statement of the distance and direction of the hole from the section. These cross-section sheets are used to show the general form and relations of the ore body and the relations of the different classes of ore with the barren areas and with each other.

A typical generalized cross section is shown in Form 5. The scale is the same as that used by the Chino company, namely, 1 inch equals 200 feet horizontally and 60 feet vertically. Each hole shows in place the number of feet of each class of ore, and of waste, that is encountered. Lines were drawn connecting points of approximately equal values; thus, by the law of averages, the relative quantities of each class were graphically shown on the section.

The figures 20, 21, 23, etc., along the top of Form 5 are the numbers of the drill holes along the line of the cross section, the cross section being constructed from the data revealed by these holes. The numbers 2' N, 3' S, etc., under some of the hole numbers mean that the hole is 2 feet north, or 3 feet south of the actual cross-section line, but the data from holes that are so close to the section are projected onto the section just as though the holes were in line. The P areas (unshaded) show the relations of waste material to ore, etc. The shaded areas that lie inside and above the heavy dotted line are ore areas, and these are divided into two different classes by two different patterns of shading. The shaded areas are further subdivided into a total of five classes of ore by light dotted lines connecting points of similar change of value in each drill hole.

The ore below the 6,050-foot level can probably not be recovered by steam-shovel mining and the present methods of rail haulage, so that the cost of recovering it will be somewhat increased.





FORM 4.—Specimen cross-section assay record of results of sinking three prospect churn-drill holes to determine copper content of ore bodies.

Elevation	HOLE A [E.100; S.0.00]				HOLE B [E.100; S.0.00]				HOLE C [E.190; S.1.30]			
	Per cent		Per cent		Per cent		Per cent		Per cent		Per cent	
	Cu <sup>a</sup>	Feet <sup>b</sup>	Cu <sup>c</sup>	Class <sup>d</sup>	Cu <sup>a</sup>	Feet <sup>b</sup>	Cu <sup>c</sup>	Class <sup>d</sup>	Cu <sup>a</sup>	Feet <sup>b</sup>	Cu <sup>a</sup>	Class <sup>d</sup>
6400	0		.00		.00		.00		.00		.00	
			.00		.00		.00		.00		.00	
			Tr		.02		.02		.02		.02	
			Tr		.06		.06		.03		.03	
			.06		.06	33	.06	P	.09	42	.09	P
		19		P	.02		.02		.10		.10	
			.03		.18		.18		.11		.11	
			.02		.30		.30		.19		.19	
30			Tr		.60		.60		.08		.08	
			.03		.90		.90		.02		.02	
			.20		.95	12	.937	L-2	.20		.20	
		9	.907	L-2	.98		.98		.42		.42	
			.95		.92		.92		.85		.85	
6350			1.03		1.10		1.10		.95		.95	
			1.10		1.12	9	1.133	F	.92	18	.882	L-1
			1.22		1.18		1.18		.08		.08	
		15	1.186	F	1.70		1.70		.75		.75	
60			1.28		1.72	12	1.750	E	.85		.85	
			1.30		1.78		1.78		1.10		1.10	
		6	.875	L-1	1.80		1.80		1.19		1.19	
			.90		2.10		2.10		1.21		1.21	
			2.00		2.22		2.22		1.30	21	1.263	G
			2.10		2.28		2.28		1.32		1.32	
			2.19		2.30	18	2.283	S	1.34		1.34	
		21	2.114	S	2.36		2.36		1.38		1.38	
			2.12		2.44		2.44		1.80		1.80	
			2.18		1.90	6	1.05	F	1.85	12	1.837	E
90			2.00		1.10		1.10		1.89		1.89	
			1.95		1.90		1.90		1.91		1.91	
			1.98		1.95		1.95		.85		.85	
6300		15	1.880	E	1.91	12	1.935	E	1.05		1.05	
			1.92		1.98		1.98		1.10		1.10	
			1.80		1.00		1.00		1.12	21	1.085	F
			1.75		.95		.95		1.08		1.08	
			.60		.40	15	.810	L-1	1.19		1.19	
			.45		.80		.80		1.21		1.21	
			.38		1.00		1.00		1.85	6	1.975	E
			.39		1.00		1.00		2.10		2.10	
120			.25		1.12	12	1.075	F	1.40	6	1.300	G
			.18		1.08		1.08		1.20		1.20	
			.12		.60		.60		.65		.65	
			.04		.32		.32		.45		.45	
		60		P	.38		.38		.30		.30	
			.03		.25		.25		.28		.28	
			.07		.20		.20		.22		.22	
			Tr		.25		.25		.20		.20	
			Tr		.20		.20		.12		.12	
6250	150		.09		.12		.12		.18	18		P
			.06		.13		.13		.14		.14	
			.04		.02		.02		.08		.08	
			.21		.08	69		P	.06		.06	
			.19		.06		.06		.04		.04	
		6	.85	L-1	.04		.04		.02		.02	
			.88		Tr		Tr		.25		.25	
			1.10		Tr		Tr		.40		.40	
			1.12		.02		.02		.85		.85	
180		15	1.48	G	.06		.06		.90	12	.912	L-2
			1.90		.06		.06		.91		.91	
			2.10		.08		.08		.99		.99	
			.70		.03		.03		1.30		1.30	
		9	.817	L-1	.07		.07		1.32	9	1.333	G
			.80		.10		.10		1.38		1.38	
6200			1.25		.40		.40		1.80		1.80	
		12	1.50	E	.80		.80		1.86	12	1.850	E
			1.25		.85		.85		1.84		1.84	
			1.64		.84	18	.861	L-1	1.90		1.90	
			1.86		.86		.86		2.10		2.10	
210			2.32		.90		.90		2.12		2.12	
			2.38		.92		.92		2.18		2.18	
			2.25		1.10		1.10		2.20		2.20	
			2.35		1.12		1.12		2.12	24	2.150	S
			2.21		1.18	15	1.204	F	2.18		2.18	
			2.29		1.30		1.30		2.14		2.14	
			2.34		1.32		1.32		2.16		2.16	
		54	2.600	S	2.00		2.00		1.80	6	1.775	E
			2.20		2.10	9	2.077	S	1.75		1.75	
240			2.22		2.13		2.13		1.20		1.20	
			2.05		.95		.95		1.18		1.18	
6150			3.10		.92	12	.950	L-2	1.16	15	1.160	F
			3.20		.98		.98		1.14		1.14	
			3.19		.95		.95		1.12		1.12	
			3.31		1.10		1.10		.65		.65	
			3.35		1.20		1.20		.78	12	.832	L-1
			1.10		1.28	18	1.266	G	.92		.92	
		12	1.150	F	1.32		1.32		.98		.98	
			1.20		1.36		1.36		1.02		1.02	
270			1.11		1.34		1.34		1.10		1.10	
			.82		1.85		1.85		1.12	18	1.120	F
			.68		1.90		1.90		1.16		1.16	
		15	.840	L-1	1.95	15	1.920	E	1.14		1.14	
			.95		1.92		1.92		1.18		1.18	
			.85		1.98		1.98		1.60		1.60	
			.90		1.10		1.10		1.85	12	1.837	E
			.52		.75		.75		1.88		1.88	
			.38		.74		.74		1.82		1.82	
6100	300		.19		.85	24	.924	L-2	1.08		1.08	
			.21		.90		.90		1.02	12	1.100	F
			.20		.95		.95		1.19		1.19	
			.10		1.20		1.20		1.11		1.11	
			.06		.90		.90		.92		.92	
		54		P	.60		.60		.96	12	.950	L-2
			.02		.40		.40		.98		.98	
			.08		.38		.38		.94		.94	
			.09		.32		.32		.65		.65	
			.01		.26		.26		.60		.60	
			Tr		.24		.24		.52		.52	
330			Tr		.10		.10		.48		.48	
			.08		.12	48		P	.39		.39	
			.20		.08		.08		.21		.21	
			.22		.06		.06		.32		.32	
			.80		.08		.08		.18		.18	
		9	.883	L-1	.12		.12		.17		.17	
			.90		.16		.16		.13	60		P
6050			1.50		.24		.24		.12		.12	
			1.55		.28		.28		.08		.08	
		12	1.700	E	.32		.32		.06		.06	
			1.70		.80		.80		.03		.03	
360			1.95		.85	12	.887	L-1	.04		.04	
			2.10		.92		.92		.07		.07	
			2.12		.98		.98		.03		.03	
		21	2.157	S	1.20		1.20		.06		.06	
			2.30		1.28		1.28		.23		.23	
			2.18		1.52	15	1.320	G	.47		.47	
			2.12		1.39		1.39		.85		.85	
			1.10		1.41		1.41		.95	9	.910	L-2
			1.00		2.00		2.00		.93		.93	
390		12	.912	L-2	2.10		2.10		1.07		1.07	
			.60		2.11	15	2.100	S	1.03		1.03	
			.95		2.19		2.19		1.17	15	1.154	F
			1.10		2.10		2.10		1.21		1.21	
6000		15	1.160	F	1.90		1.90		1.29		1.29	
			1.15		1.85		1.85		1.85		1.85	
			1.30		1.70	12	1.775	E	1.83		1.83	
			1.20		1.65		1.65		1.82	15	1.880	E
			.80		.95	9	.950	L-2	1.93		1.93	
		12	.838	L-1	.90		.90		1.97		1.97	
			.82		.60		.60		2.10		2.10	
420			.60		.55		.55		2.13		2.13	
			.62		.45		.45		2.17		2.17	
			.61		.38		.38		2.19	21	2.128	S
			.59		.32		.32		2.21		2.21	
			.58		.28		.28		2.03		2.03	
			.42		.14		.14		2.07		2.07	
			.38		.11		.11		1.65		1.65	
			.36		.10		.10		1.63		1.63	
5950	450	60		P	.00	57		P	1.61	18	1.600	E
			.12		.04		.04		1.63		1.63	
			.18		.01		.01		1.67		1.67	
			.08		.08		.08		.62		.62	
			.02		.09		.09		.58		.58	
			.02		.							

The notation 10 N., East and West crosssection, at the top of the cross section means that this is an east and west cross section, and is the tenth north from cross section 1. The figures 1000, 1500, etc., are the horizontal distances in feet. The vertical distances, referred to sea level, are shown by the figures 6050, 6200, 6350, etc., on the right margin.

The use of colors to denote the different classes of ores shown by the cross section adds greatly to its clearness. In the form as used at the mine two colors, indicated by shading in the form as presented, were used to bring out the special relations.

Figure 2 represents two drill holes. Hole 1 shows 3 feet of class E ore below 17 feet of barren rock. Hole 2 shows 10 feet of class E ore

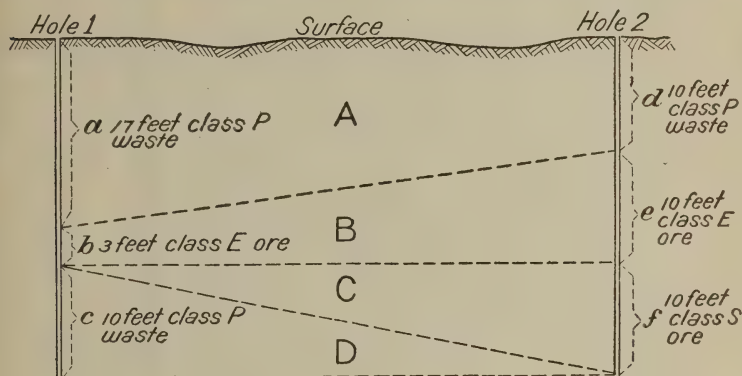


FIGURE 2.—Method of estimating classes of ore from drill records so as to show them on cross sections. *a*, hole 1, 17 feet of class P rock or waste; *b*, 3 feet of class E ore; *c*, 10 feet of class P rock; *d*, hole 2, 10 feet of class P rock; *e*, 10 feet of class E ore; *f*, 10 feet of class S ore. The area A would be waste, the area B would be class E ore, and the triangular area C would be S ore. It would be figured as extending up to hole 1 though no S ore had actually been encountered in that hole. The D area would be waste rock that would not be moved.

and 10 feet of class S ore below 10 feet of barren rock. A straight (dotted) line connects the top boundary of class E ore in hole 1 with the top boundary of this class of ore in hole 2; and similarly for the bottom boundary. Although no class S ore was found in hole 1 yet the class S ore body found in hole 2 may maintain its full thickness out nearly to hole 1, so, by the law of averages, it is approximately correct to figure the ore as occupying the triangular space C.

Cross sections are very helpful in showing the trend of ore bodies and their form and general relations. Where geological conditions can be shown on the cross sections they are especially valuable in studying ore deposits. Such cross sections as are presented are not quite accurate enough for use in making final estimates of ore tonnages. A more accurate method of ore estimation is discussed in a subsequent section.



**GENERAL CONSIDERATIONS IN DELIMITING ORE BODIES.**

An inspection of the completed cross-section sheets made fairly evident just what ground could profitably be mined under the conditions. With the cross-section data as a basis, the limiting outlines of the ore bodies were drawn on maps showing the surface configuration, drill holes, etc. The chief considerations that governed the mapping of the horizontal limits to which ore could be mined were as follows: (1) The depth of the overburden and the slopes at which its banks would stand; (2) the local size and thickness of the ore body and its character and value; (3) the method of mining—whether the ore body could all be taken out by steam shovels at about the same average cost as the main part of the ore body, or whether extra expense would be involved.

Of course, it was recognized that there must necessarily be a limit to the depth to which steam-shovel mining with locomotive haulage could be carried. After consideration of various factors it was believed that below the 6,050-foot level (measured from sea level) the great open-cut mining pits would be so deep and so restricted in area at the bottom that certain modifications in the methods of mining the ore below that level—a relatively small percentage of the total ore—would have to be adopted. The chief factors regulating the depth to which it would be convenient to mine with steam shovels were believed to be as follows: (1) The horizontal extent of the ore body, in relation to its depth, would have to be considered because a certain amount of "elbow room" would be necessary for steam shovels, car tracks, etc.; (2) the maintenance of economical transportation grades for hauling out the ore and waste would limit the depth of working by steam shovels and locomotive haulage; (3) if the horizontal dimensions of the deposit were great enough, steam shovels might load the ore even in very deep pits, and it might be hauled out by means of Shay or rack-gear locomotives, or over inclined planes with cable haulage.

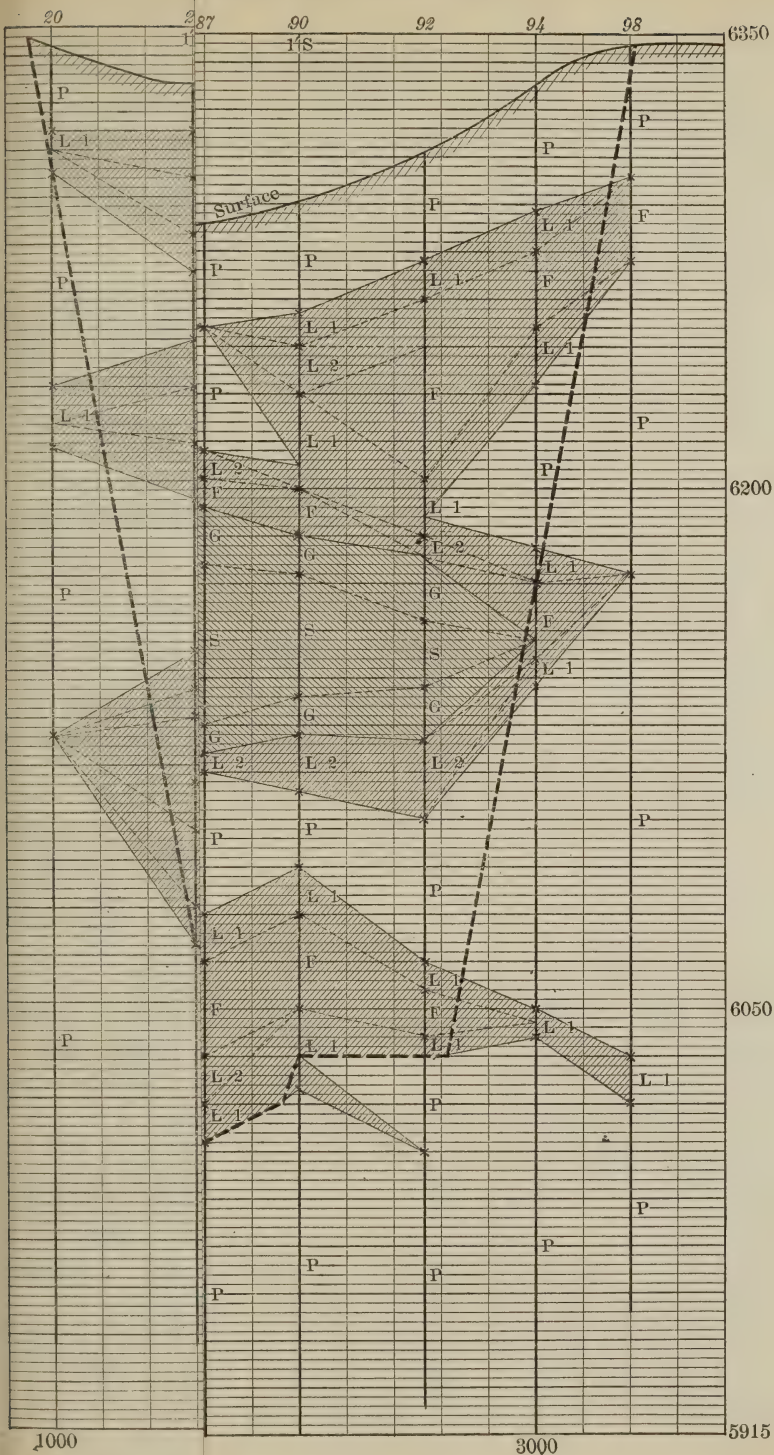
Estimates of the ore above the 6,050-foot level were kept separate from those of the relatively small proportion of ore below that level, as the cost of taking out the lower ore would be slightly greater.

**METHODS OF ESTIMATING ORE.**

When the facts collected in prospecting and sampling the ore bodies were supplemented by geologic and by general mining cost data, the whole body of information thus systematically gathered was used in estimating the tonnage of rock rich enough to be classed as ore and available for mining.



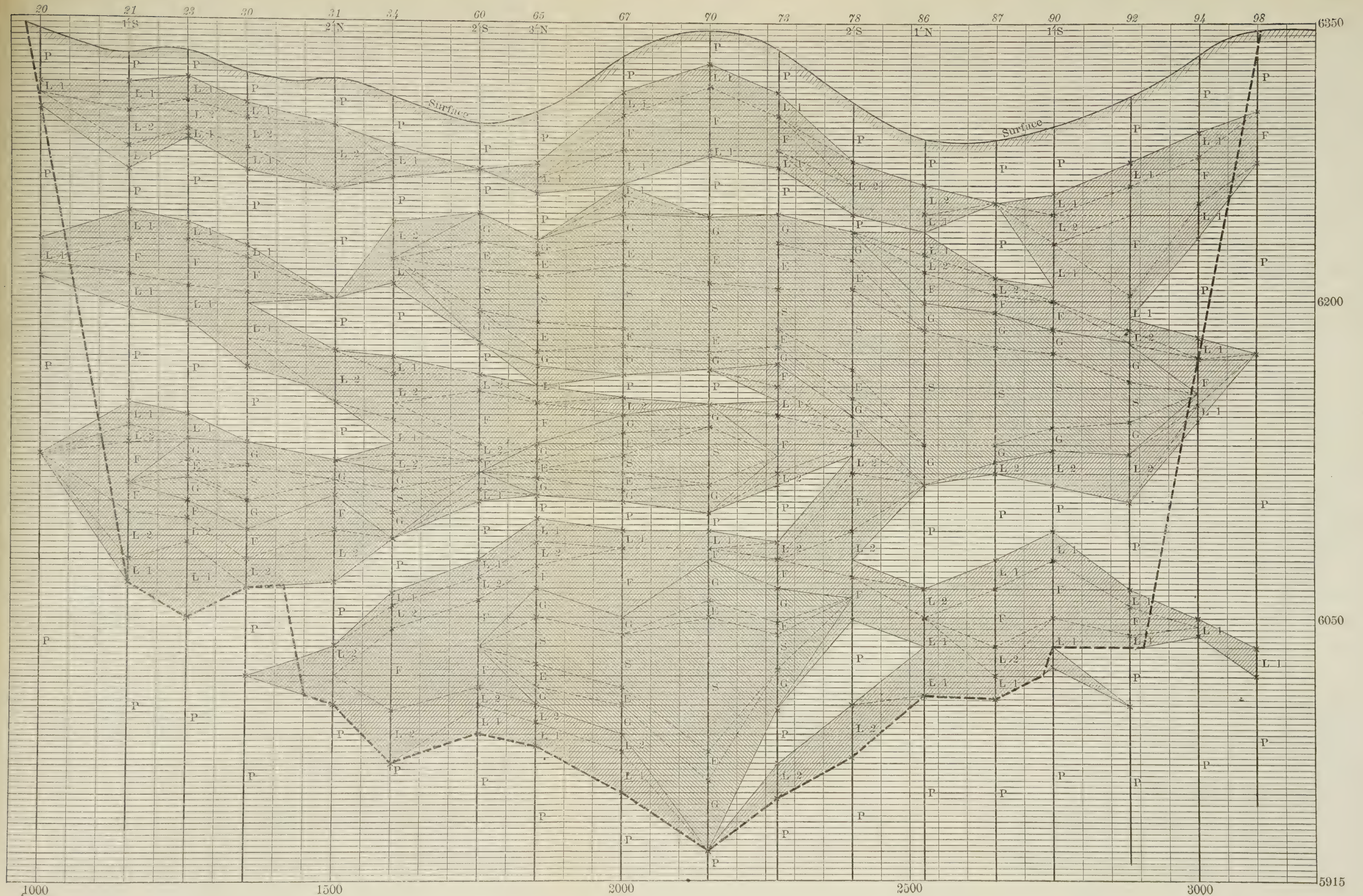
grades of ore, are explained on page 15.]





FORM 5.—Typical cross section of ore body.

[This section was designated "Cross Section 10N, East and West." One inch represents 200 feet horizontally and 60 feet vertically, Letters, P, L1, L2, F, G, E, S, showing grades of ore, are explained on page 15.]





### ASSAY PLAN.

In proceeding toward a methodical solution of the problem of ore estimation the first step was to make an assay plan. This consisted of a copy of the original drilling map showing each hole, by number, its location, the number of feet of ore encountered in it, the average value of the ore, the number of feet of waste that would have to be moved from the ore there, etc.

### PRELIMINARY ESTIMATES.

Some general idea of the form and size of the ore body, as well as possible methods of working it, had to be obtained before definite figures on the amount of minable ore available could be made. For instance, if only a quarter of a million tons of ore was in the deposit, the average cost per ton for mining would be more, other things being equal, than if it contained 5,000,000 tons.

After determination had been made, in a general way, that the ore body was properly situated and extensive enough to warrant mining it with steam shovels on a large scale, an estimate was made of the probable cost of removing and dumping the valueless stripping material and mining the ore by open-cut methods on a scale commensurate with the great size of the deposit. The estimated cost, including concentrating, smelting, marketing the product, all wastage, and all expenses was \$2 per ton for moist ore. From the cost data on mining, shipping, and treating the ore from the somewhat similar ore bodies of the Utah Copper Co.'s properties, it was known that the figure mentioned was a conservative one on which to base estimates of the ore reserves.

In order to facilitate the estimation of ore, the critical worth, below which mining at a profit could not in general be counted on, was expressed in units or values of copper. At 13 cents per pound for copper, ore that assayed 0.8 per cent, or 16 pounds of metal per ton, valued at \$2.08, meant at least 8 cents per ton profit, if, as above stated, all costs and wastage allowances amounted to \$2 per ton.

After figures of this kind had been prepared in this general way, it was decided to adopt 0.8 per cent of copper as what might be called the critical content. All rock containing a less percentage could not on this basis be estimated as ore. However, it was realized that if the price of copper went up considerably, then wherever practicable much of the leaner rock would undoubtedly be mined as ore, but such a contingency could not, of course, be made a factor in a conservative estimate. Wherever a mass of lean rock happened to be surrounded by ore, so that separating it was unprofitable, then of course it was figured in as a diluent, which reduced somewhat the average tenor of the ore.



A critical copper content of 0.8 per cent having been fixed as the content below which the rock could not conservatively be called ore, there being very little gold or silver in it, the estimate was made first by the cross-section method, which consisted in estimating the mass of ore by means of square prisms and then by figuring its value from the drill-hole and cross-section data. Later, as a check, the more exact triangular-prism method was used. As this method is better adapted for close and careful ore estimation, though involving about the same general principle as the cross-section method, it is explained in detail below.

#### ESTIMATING ORE BY TRIANGULAR-PRISM METHOD.

From the assay record of each drill hole the ore reserves were calculated. For these calculations three well-bound books with blank pages were used. They were each about  $9\frac{1}{2}$  by 12 inches in size and were cross ruled on a scale of 4 spaces to the inch. Book 1 may be called the log book of prospect drill holes. On page 1 were recorded the ore calculations for hole 1, on page 2 the calculations for hole 2, and so on.

The following examples are to show how the drill-hole records, prepared on Form 4, were calculated in Book 1:

*Specimen calculation sheets from drill-hole records shown in Form 4.*

LOG OF HOLE.				HOLE A (E. 100; S. 0.00.).			ORE OF CLASS L.2.		
Feet of ore.	Class.	Per cent copper.	Foot per cent.	Feet of ore.	Per cent copper.	Foot per cent.	9	.907	8.16
19	P <sup>a</sup>	.....	.....	21	2.114	44.39	12	.912	10.94
9	L 2	.907	8.16	51	2.600	132.60	21		19.10
15	F	1.186	17.79	21	2.157	45.30	ORE OF CLASS L.1.		
6	L 1	.875	5.25	27	2.135	57.64	6	.875	5.25
21	S	2.114	44.39	120		279.93	6	.850	5.10
15	E	1.880	28.20	ORE OF CLASS E.			9	.817	7.35
60	P <sup>a</sup>	.....	.....	15	1.880	28.20	15	.840	12.60
6	L 1	.850	5.10	12	1.500	18.00	9	.883	7.94
15	G	1.480	22.20	12	1.700	20.40	12	.838	10.05
9	L 1	.817	7.35	39		66.60	9	.850	7.65
12	E	1.500	18.00	ORE OF CLASS G.			66		55.94
51	S	2.600	132.60	15	1.480	22.20	CLASS P WASTE.		
12	F	1.150	13.80	ORE OF CLASS F.			193	.....	.....
15	L 1	.840	12.60	15	1.186	17.79	520		519.99
54	P <sup>a</sup>	.....	.....	12	1.150	13.80			
9	L 1	.883	7.94	15	1.160	17.40			
12	E	1.700	20.40	12	1.158	13.89			
21	S	2.157	45.30	12	1.112	13.34			
12	L 2	.912	10.94	66		76.22			
15	F	1.160	17.40						
12	L 1	.838	10.05						
60	P <sup>a</sup>	.....	.....						
12	F	1.158	13.89						
27	S	2.135	57.64						
12	F	1.112	13.34						
9	L 1	.850	7.65						
520			519.99						

<sup>a</sup> Waste.

## HOLE B (E. 210; S. 0.00.).

## LOG OF HOLE.

Feet of ore.	Class.	Per cent copper.	Foot per cent.
33	P <sup>a</sup>	.....	.....
12	L 2	.937	11.24
9	F	1.133	10.19
12	E	1.750	21.00
18	S	2.283	41.09
6	F	1.050	6.30
12	E	1.935	23.22
15	L 1	.810	12.15
12	F	1.075	12.90
69	P <sup>a</sup>	.....	.....
18	L 1	.861	15.49
15	F	1.204	18.06
9	S	2.077	18.69
12	L 2	.950	11.40
18	G	1.266	22.78
15	E	1.920	28.80
24	L 2	.924	22.18
48	P <sup>a</sup>	.....	.....
12	L 1	.887	10.64
15	G	1.320	19.80
15	S	2.100	31.50
12	E	1.775	21.30
9	L 2	.950	8.55
57	P <sup>a</sup>	.....	.....
12	L 1	.850	10.20
9	F	1.233	11.10
15	S	2.100	31.50
6	G	1.300	7.80
9	L 1	.883	7.95
528			435.83

<sup>a</sup> Waste.

## ORE OF CLASS S.

Feet of ore.	Per cent copper.	Foot per cent.
18	2.283	41.09
9	2.077	18.69
15	2.100	31.50
15	2.100	31.50
57		122.78

## ORE OF CLASS E.

12	1.750	21.00
12	1.935	23.22
15	1.920	28.80
12	1.775	21.30
51		94.32

## ORE OF CLASS G.

18	1.266	22.78
15	1.320	19.80
6	1.300	7.80
39		50.38

## ORE OF CLASS F.

9	1.133	10.19
6	1.050	6.30
12	1.075	12.90
15	1.204	18.06
9	1.233	11.10
51		58.55

## ORE OF CLASS L 2.

Feet of ore.	Per cent copper.	Foot per cent.
12	.937	11.24
12	.950	11.40
24	.924	22.18
9	.950	8.55
57		53.27

## ORE OF CLASS L 1.

15	.810	12.15
18	.861	15.49
12	.887	10.64
12	.850	10.20
9	.883	7.95
66		56.43

## CLASS P WASTE.

207	.....	.....
528		435.83

## HOLE C (E. 190; S. 130.).

LOG OF HOLE.				ORE OF CLASS S.			ORE OF CLASS L2.		
Feet of ore.	Class.	Per cent copper.	Foot per cent.	Feet of ore.	Per cent copper.	Foot per cent.	Feet of ore.	Per cent copper.	Foot per cent.
42	Pa	.....	.....	24	2.150	51.60	12	.912	10.94
18	L 1	.882	15.88	21	2.128	44.69	12	.950	11.40
21	G	1.263	26.52				9	.910	8.19
12	E	1.837	22.04	45		96.29	6	.900	5.40
21	F	1.085	22.79						
6	E	1.975	11.85	ORE OF CLASS E.			39		35.93
6	G	1.300	7.80	12	1.837	22.04			
48	Pa	.....	.....	6	1.975	11.85	ORE OF CLASS L1.		
12	L 2	.912	10.94	12	1.850	22.20	18	.882	15.88
9	G	1.333	12.00	6	1.775	10.65	12	.832	9.98
12	E	1.850	22.20	12	1.837	22.04			
24	S	2.150	51.60	15	1.880	28.20	30		25.86
6	E	1.775	10.65	18	1.600	28.80	CLASS P WASTE.		
15	F	1.160	17.40				180	.....	.....
12	L 1	.832	9.98	81		145.78	531		495.52
18	F	1.120	20.16	ORE OF CLASS G.					
12	E	1.837	22.04	21	1.263	26.52			
12	F	1.100	13.20	6	1.300	7.80			
12	L 2	.950	11.40	9	1.333	12.00			
60	Pa	.....	.....	39	1.397	54.48			
9	L 2	.910	8.19						
15	F	1.154	17.31	75		100.80			
15	E	1.880	28.20	ORE OF CLASS F.					
21	S	2.128	44.69	21	1.085	22.79			
18	E	1.600	28.80	15	1.160	17.40			
30	Pa	.....	.....	18	1.120	20.16			
6	L 2	.900	5.40	12	1.100	13.20			
39	G	1.397	54.48	15	1.154	17.31			
531			495.52						
a Waste.				81		90.86			

In explanation of the calculation sheets shown it may be said that the number of the hole and its location by coordinates are set down at the top of the page, as Hole A (E. 100; S. 0.00). In the left-hand column the number of feet of each class of ore or waste, according to the classification mentioned on page 15, is shown in proper relative position, having been taken direct from the assay record of the drill hole (Form 4). The average assay of each class of ore in percentage of copper is shown in the second column.

Now 10 feet of 1 per cent ore and 20 feet of 2 per cent ore would not give 30 feet of 1.5 per cent ore, but 30 feet of 1.667 per cent ore. This correct average is obtained by taking account of the fact that there is twice as much of the rich 2 per cent ore as of the 1 per cent ore. The calculation follows:

Feet of ore.		Per cent copper.		Foot per cent.
10	×	1	=	10
20	×	2	=	40
30				50

Dividing the total foot per cent, 50, by the total feet of ore, 30, gives 1.667 per cent of copper as the average value throughout the 30 feet.



In like manner the number of feet of each class of ore is multiplied by its assay value, and the product is set down under the column marked "Foot per cent." The depth of the hole to the bottom of the lowest ore divided into the total feet per cent would give the average value for that hole. On the right-hand side of the page are segregated the figures for the different classes of ore for the given hole, the number of feet of each class, and its foot per cent.

Book 2, used in the ore calculations, may be called the "triangular-prism book." In its records the ore body is regarded as composed of a series of vertical triangular prisms, and the ore grade of each prism is separately calculated. The first step in this calculation is to take a copy of the drill-hole map and connect, by straight lines, each drill hole with its nearest neighbors, in such a way that the whole ore-bearing area is divided into triangles, with the apices of each triangle, or rather the edges of each triangular prism, marked

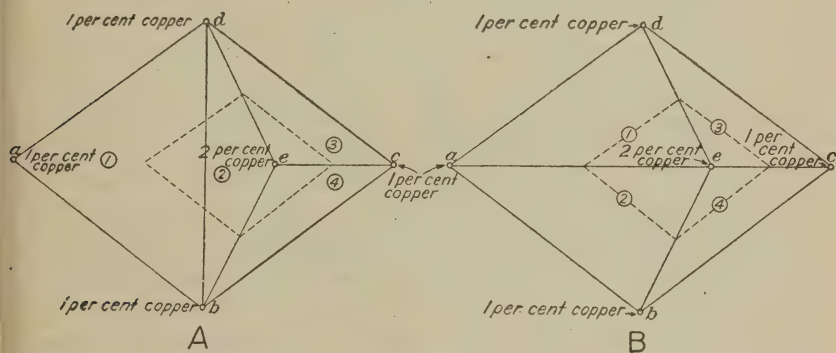


FIGURE 3.—Methods of dividing ore area into triangles. A, incorrect method; B, correct method.

by a drill hole. The ore grade in each triangular prism is thus closely approximated by the average grades found in the three drill holes forming its edges.

There is a right way and a wrong way to lay off these triangles. Suppose five drill holes were placed and showed values about as in figure 3.

Figure 3, A, shows the wrong way to triangulate these holes, if divided in the way shown, triangle 1 would show only 1 per cent of ore, whereas in reality the 2 per cent ore should be calculated as extending halfway from *e* to *a*, as indicated by the dotted lines. Of course, in any particular case, the rich ore might not extend halfway toward *a*, but, according to the law of averages, it should be calculated as though it did, so as to make proper apportionment of the high as well as of the low values. If the drill-hole area were divided as in figure 3, A, triangle 1 would be undervalued and triangle 2 overvalued.

Correct apportionment of the values to each of the triangles is shown in figure 3, B. The data on which to base properly evaluated

triangulation are obtained not only from the average assays of each hole, but largely from the geologic information, such as the continuity of the ore values, shown on the cross sections.

When all of the ore-bearing drill holes have been connected by lines so that each hole forms the apex of one or more triangles, each triangle so formed is given a number. On page 1 of the triangle book are found the calculations for triangle 1, as follows:

*Specimen page from triangle book.*

**TRIANGLE 1. AREA, 7,150 SQUARE FEET. HOLES A, B, AND C.**

[Data taken from calculations on pages 20, 21, and 22.]

Hole.	Class of ore.												
	S.		E.		G.		F.		L 2.		L 1.		P. feet of waste.
	Feet of ore.	Foot per cent.	Feet of ore.	Foot per cent.	Feet of ore.	Foot per cent.	Feet of ore.	Foot per cent.	Feet of ore.	Foot per cent.	Feet of ore.	Foot per cent.	
A.....	120	279.93	39	66.60	15	22.20	66	76.22	21	19.10	66	55.94	193
B.....	57	122.78	51	94.32	39	50.38	51	58.55	57	53.37	66	56.43	207
C.....	45	96.29	81	145.78	75	100.80	81	90.86	39	35.93	30	25.86	180
a.....	222	499.00	171	306.70	129	173.38	198	225.63	117	108.40	162	138.23	580
b.....	74	166.33	57	102.23	43	57.79	66	75.21	39	36.13	54	46.08	193.33
c.....	529,100		407,550		307,450		471,900		278,850		386,100		.....
d.....	1,189,260		730,945		413,199		537,752		258,330		329,472		.....

a Line a shows sum of the number of feet and foot per cent of the different classes of ore for the three holes.

b Line b shows one-third of a, or the average of the three holes.

c Line c shows average thickness multiplied by area, giving cubic feet of each class of ore in prism.

d Line d shows average foot per cent multiplied by area giving cubic foot per cent or "volume assay" of each class of ore in prism.

P.

$$\begin{array}{r} \text{A} = 193 \\ \text{B} = 207 \\ \text{C} = 180 \\ \hline 580 \end{array} \quad \frac{580}{3} \times 7,150 = \frac{4,147,000}{3} = 1,382,333 \text{ cubic feet, or amount of waste contained in triangular prism 1.}$$

The sample of the calculations set forth in the triangle book shows that the area of the triangle and the number of the holes from which its ore content is calculated are set down at the top of the page. Line a shows the total number of feet of ore and the foot per cent of each class of ore in each hole. The average thickness and the average foot per cent of each class for the prism (one-third of a) are entered in line b. Line c shows the average thickness of each class of ore multiplied by the cross-sectional area of the prism. This gives the number of cubic feet—the volume—of a particular class of ore. The d line shows the cubic foot per cent, or the volume assay, for each class of ore in the triangular prism. It is obtained by multiplying the average foot per cent by the cross-sectional area of the prism.

The third book for ore calculations is of the same type as the other two, and may be called the book of ore totals. In it are recorded the volume of ore and the volume assay, or cubic feet per cent, of each class in each triangular prism for each ore body. Let us

assume that ore body A contains 100 triangular prisms of ore, each exactly equal to prism 1, already recorded. The ore in such an ore body would be summed up in the book of ore totals somewhat as follows:

*Specimen page from book of ore totals.*

**TOTAL ORE IN ORE BODY A.**

Class of ore.	Cubic feet of ore.	Volume assay or cubic feet per cent.	Per cent copper.	Tons of ore, at 13 cubic feet per ton.
S.....	52,910,000	118,926,000	2.247	4,070,000
E.....	40,755,000	73,094,500	1.793	3,135,000
G.....	30,745,000	41,319,900	1.343	2,365,000
F.....	47,190,000	53,775,200	1.139	3,630,000
L 2.....	27,885,000	25,833,000	.926	2,145,000
L 1.....	38,610,000	32,947,200	.853	2,970,000
	238,095,000	345,895,800	1.453	18,315,000

Dividing the volume assay, or cubic feet per cent, of ore by the cubic feet gives the average assay value, in per cent, for each class and for the total ore.

In order to convert the cubic feet of ore into tons, the average specific gravity of the ore body must be determined.<sup>a</sup> If found to be 2.5, then a cubic foot of ore would weigh 2.5 times the weight of a cubic foot of water, or  $2.5 \times 62.5 = 156.25$  pounds; thus 13 cubic feet of such ore would weigh approximately a ton. Thirteen cubic feet of Chino ore in the solid and 21 cubic feet of broken ore were found to average practically a ton, and this average was used in the calculation of ore reserves. Dividing the cubic feet of ore by 13 gives the tonnage shown above—18,315,000 tons of ore, containing an average assay value of 1.453 per cent copper, or practically 266,117 tons of copper. The actual amount of this copper that could be recovered would of course depend on the percentage of recovery made by the mill and smelter.

To summarize, holes were drilled through the ore body every 100 or 200 feet, and these were sampled at depth intervals of 3 feet. The assay values from each sample were arranged in order, and grouped into classes as shown in Form 4. The classes were entered in the log book of prospect drill holes, and the drill holes were grouped together in sets of threes, each set of three marking the edges of a vertical triangular prism. The average values for each prism were calculated in the triangular prism book; and finally the total ore for each ore body was summed up in the book of total ore. These books, with all calculations, are preserved with other company records.

<sup>a</sup> For diagram and formula convenient for ore calculations see Mead, W. J., The relation of density, porosity, and moisture to the specific volume of ores: Econ. Geol., vol. 3, June-July, 1908, pp. 319-325.



The results from the book of ore totals were arranged into tables as follows: A table of total ore developed up to March 1, 1912, and the yardage of stripping that would have had to be moved to recover it; a table of the total steam-shovel ore developed up to the same date, and the yardage of stripping that covered it; a table of additional steam-shovel ore developed after March 1, 1912. Of these tables the one covering the amount of steam-shovel ore developed up to the date mentioned follows:

Data on amount of steam-shovel ore developed up to Mar. 1, 1912.

Name of ore body prospected.	Steam-shovel ore.												Stripping. b		Ratio of ore to waste.				
	Class S. c		Class E. c		Class G. c		Class F. c		Class L2. c		Class L1. c		Class P. c			Slope.	Cubic yards broken.	Tons.	
	Tons.	Aver- age-as- say a	Tons.	Aver- age assay.	Tons.	Aver- age assay.	Tons.	Aver- age assay.	Tons.	Aver- age assay.	Tons.	Aver- age assay.	Total tons.	Aver- age assay.					
C. Y. B. ....	3,441,710	2.85	2,617,770	1.67	1,758,932	1.37	2,084,220	1.13	1,034,788	0.95	625,754	0.86	458,342	0.55	12,021,516	1.72	6,527,129	8,392,023	1:0.70
R. T. W. ....	4,681,412	2.91	3,320,939	1.71	2,219,884	1.37	3,282,042	1.11	1,690,213	.94	1,240,122	.85	916,712	.58	17,371,323	1.68	10,989,870	14,129,833	1:0.81
Hearst. ....	13,324,098	3.41	4,926,392	1.66	4,429,134	1.35	4,110,055	1.10	2,784,595	.94	2,353,135	.84	1,512,191	.84	61,337,439	2.08	21,028,461	27,036,593	1:0.81
Old Fort. ....	31,195	3.30	61,913	1.59	133,116	1.32	76,826	1.12	31,464	.91	6,788	.82	19,016	.48	360,918	1.41	825,180	418,088	1:1.16
Sierra. ....	1,930,258	2.71	4,058,965	1.70	2,938,513	1.35	2,895,046	1.10	1,440,814	.93	994,511	.85	699,034	.63	14,957,741	1.46	13,229,328	17,009,136	1:1.14
Estrella. ....	1,120,984	2.66	1,206,484	1.69	555,521	1.35	858,734	1.10	210,882	.94	193,593	.85	242,991	.51	4,389,190	1.64	9,367,943	12,044,498	1:2.74
Total. ....	24,529,657	3.14	16,192,463	1.68	12,035,100	1.36	13,306,923	1.11	7,192,756	.94	5,433,908	.85	3,849,486	.59	82,540,287	1.81	61,467,911	79,030,171	1:0.96

<sup>a</sup> Each average assay is expressed as per cent of copper per ton of ore.<sup>b</sup> The slope at first was  $\frac{1}{2}$  horizontal to 1 vertical, but in 1915, after the stripping had been nearly completed, a slope of  $\frac{1}{4}$  to 1 was adopted.<sup>c</sup> For explanation of ore classification, see p. 15.

## FORMULAS FOR ESTIMATING ORE.

## FORMULA FOR DETERMINING LOWEST GRADE OF ROCK THAT MAY BE CLASSED AS ORE.

The formulas following may be largely credited to Mr. L. E. Foster, assistant superintendent of the Chino company's mine. The authors merely developed a little further his fundamental formula. Formula 1 is for use in determining what grade of rock might be mined as ore under varying conditions of costs and of copper prices. The factors in the equation are explained below. Certain cost figures are given to be substituted in the equations. These figures, however, are being modified by more efficient methods as time goes on.

$$(1) \quad \frac{M+F_0+m}{AC \times 2000 \times 0.95} + \frac{f_0+S}{D \ 2000 \times 0.95} + R+E+c = B \frac{100}{105}$$

Simplifying equation 1 we get:

$$(2) \quad \frac{M+F_0+m}{1900 \ C} + \frac{A \ (f_0+S)}{1900 \ D} + A(R+E+c) = AB \frac{100}{105}$$

Transposing equation 2 gives:

$$(3) \quad \frac{M+F_0+m}{1900 \ C} = A \left[ \frac{100 \ B}{105} - \frac{(f_0+S)}{1900 \ D} - (R+E+c) \right]$$

Transposing equation 3 gives:

$$(4) \quad A = \frac{\frac{M+F_0+m}{1900 \ C}}{\frac{100 \ B}{105} - \frac{(f_0+S)}{1900 \ D} - (R+E+c)}$$

Simplifying equation 4 gives:

$$(5) \quad A = \frac{M+F_0+m}{1900 \ C \left[ \frac{100 \ B}{105} - \frac{(f_0+S)}{1900 \ D} - (R+E+c) \right]}$$

*Values substituted for factors in foregoing equations.*

Factor.	Definition of factors.	Value.
M=	Mining costs per dry ton of ore.....	\$0.4683
F <sub>0</sub> =	Freight from mine to mill per dry ton of ore.....	.1111
m=	Milling costs per dry ton of ore.....	.5776
f <sub>0</sub> =	Freight on concentrates per dry ton from mill to smelter.....	1.1191
S=	Smelting charges per dry ton of concentrates.....	6.352
R=	Refining and delivering charges per pound of copper paid for.....	.0106
E=	Export freight and insurance charges per pound of copper paid for.....	.0014
c=	Commission on sales.....	$\frac{1}{100}B$
B=	Value per pound of copper.....	.13
C=	Per cent of extraction by the mill.....	65
D=	Per cent of copper in the concentrates.....	25
A=	Per cent of copper per dry ton of ore.	



The constant 2,000 stands for the 2,000 pounds in each ton of ore. The constant 0.95 stands for the 95 per cent of the copper content, or the net copper that the smelter actually pays for. The dry ton of ore figured in these calculations is the ore with the weight of its moisture content deducted. The moisture content in the ore averages about 10 per cent.

About eight months elapse from the time the ore is mined until the copper is sold. During this period capital is tied up in the metal. Interest on this capital is allowed at the rate of 5 per cent for the eight months. This interest factor is introduced into the equation by multiplying B, the value of copper, by  $\frac{100}{105}$ , a fraction that represents the lessened value of money by reason of the interest deduction.

In the expression  $AC \times 2000 \times 0.95$ , used as denominator in the first member of equation 1, 2,000 pounds of ore multiplied by A, the percentage of copper contained in the ore expressed decimally, multiplied by C, the percentage of copper that is saved by the mill, multiplied by 0.95, the percentage of the copper from the mill that is paid for by the smelter, gives the net amount of copper in pounds recovered from each ton of ore. The expression  $\frac{M + F_0 + m}{AC \times 2000 \times 0.95}$ , means simply the net number of pounds of copper recovered, divided into the cost of recovering it. In the same way,  $\frac{f_0 + S}{2000D \times 0.95}$  means the net copper recovered from the concentrates at the smelter divided into the cost of recovering it. The whole equation then represents the total net cost of copper with the different factors expressed as mathematical quantities, equated to  $B \frac{100}{105}$ , which is the value of the copper lessened by an interest charge of 5 per cent. As the equation stands, nothing is charged for the interest lost on the deferred profits. In general, application of the formula interest on the capital tied up in the enterprise should be charged.

By substituting in equation 5 the different costs selected as being well within the factor of safety for the first estimate of the ore values, and simplifying, we get:

$$(6) \quad A = \frac{1.157}{1235 (0.1238 - 0.0157 - 0.0133)} = \frac{1.157}{1235 (0.0948)} = \frac{1.157}{117.078} \\ = 0.00988, \text{ or } 0.988 \text{ per cent.}$$

Now by eliminating M, the cost of mining, from equation 5 and then solving for A, we find what low-grade material, which had to be removed in order to get at richer ore, could profitably be sent to the mill rather than to the waste dump. The figures follow.

$$(7) \quad A = \frac{0.6887}{117.0533} = 0.005882, \text{ or } 0.5882 \text{ per cent.}$$

Equation 7 is equation 6 with the M values—the cost of mining—eliminated. Some rock that contains between 0.5 and 0.8 per cent of copper has to be moved to get at good ore; the cost of sending it to the dump is about the same as that of sending it to the mill. The question to be solved under such conditions is: Having this material mined and loaded because of the necessity of getting it out of the way, what content of copper will decide whether it shall be sent to the mill or to the waste dump. Equation 7 gives the answer—all rock that contains more than 0.5882 per cent of copper should, under the conditions given, go to the mill, and all rock that contains a lesser proportion of copper should go to the waste dump.

FORMULA TO DETERMINE WHETHER ORE SHOULD BE SMELTED IN THE CRUDE, OR MILLED AND THE CONCENTRATES SMELTED.

The formula used to determine what grade of ore from the mine should be shipped direct to the smelter, or to show where the cleavage between smelting ore and concentrating ore should be, is discussed below. When the total cost per pound of copper paid for is the same, whether the ore be smelted in the crude or milled and the concentrates smelted, then the following equation holds:

$$(8) \quad \frac{M+F_s+S_o}{Ay} + R+E+c = \frac{M+F_o+m}{ACy} + \frac{f_o+S_e}{Dy} + R+E+c.$$

The values substituted in this formula are in general fairly representative of the costs at Santa Rita up to the middle of 1914. Since then improvements have been made.

*Values substituted for factors in equation 8.*

Factor.	Definition of factor.	Value.
M	=Mining cost per dry ton of ore.....	\$0.4683
F <sub>s</sub>	=Freight on crude ore from mine to smelter per dry ton.....	1.1111
F <sub>o</sub>	=Freight on ore from mine to mill per dry ton.....	.1111
m	=Milling costs per dry ton of ore.....	.5776
f <sub>o</sub>	=Freight on concentrates, mill to smelter, per dry ton.....	1.1191
S <sub>o</sub>	=Smelting charges for crude ore per dry ton.....	7.25
S <sub>e</sub>	=Smelting charge per dry ton of concentrates.....	6.352
R	=Refining and delivering per pound of copper paid for.....	.0106
E	=Export freight, insurance, etc., per pound of copper paid for.....	.0014
c	=Commission.....	$\frac{1}{100}$ B
B	=Value per pound of copper.	
C	=Percentage of extraction by mill.	
D	=Percentage of copper in the concentrates.	
A	=Percentage of copper per ton of ore (expressed decimally).	
y	=(2000 multiplied by 0.95)=1900.	
0.95	=Percentage of the copper content which is recovered by the smelter (expressed decimally).	

Multiplying equation 8 by  $A_y$  we get:

$$(9) \quad M + F_s + S_o + A_y(R + E + c) = \frac{M + F_o + m}{C} + \frac{A(f_o + S_o)}{D} + A_y(R + E + c)$$

Transposing:

$$(10) \quad \frac{A(f_o + S_o)}{D} + A_y(R + E + c) - A_y(R + E + c) = M + F_s + S_o - \left( \frac{M + F_o + m}{C} \right)$$

Hence:

$$(11) \quad A = \frac{\frac{M + F_s + S_o}{1} - \frac{(M + F_o + m)}{C}}{\frac{f_o + S_o}{D}} = \frac{D[C(M + F_s + S_o) - (M + F_o + m)]}{C(f_o + S_o)}$$

If the value of the mill recovery (C) and the percentage of copper the concentrates contain (D) are known, these values can be substituted in equation 11 and the value of A obtained. Below are a few values of A for given values of C and D.

*Values of A for given values of C and D.*

C.	D	A	C	D	A	C	D	A
<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>
60	20	18.47	60	25	23.09	60	30	27.71
65	20	18.87	65	25	23.59	65	30	28.31
70	20	19.21	70	25	24.01	70	30	28.82

In other words, with C at 60 per cent and D at 20 per cent, the rock would have to contain 18.47 per cent of copper before the cost of recovering that copper would be the same whether the ore was smelted in the crude or milled and the concentrates smelted. For given values of C and D, the corresponding values for A represent critical values below which the ore should be milled and the concentrates smelted and above which it should be shipped direct to the smelter.

When the profits on every dry ton of ore are equal whether the ore is smelted in the crude or milled and the concentrates smelted, then the following is true: The value of the copper recovered from each dry ton of ore by direct smelting ( $VCu_s$ ) minus the total cost of recovering it ( $C_s$ ) is equal to the value of the copper recovered from each dry ton of ore by milling and smelting the concentrates ( $VCu_{m+s}$ ) minus the cost of all the expenses of treating and handling each ton of ore ( $C_{m+s}$ ). Putting this in equational form:

$$(a) \quad VCu_s - C_s = P_s = \text{profits from direct smelting.}$$

$$(b) \quad VCu_{m+s} - C_{m+s} = P_{m+s} = \text{profits from milling and smelting the concentrates.}$$

But if  $P_s = P_{m+s}$ , then

$$(c) \quad VCu_s - C_s = VCu_{m+s} - C_{m+s}.$$



Now, substituting in equation *c* the mathematical expressions for the value of the net copper extracted from each ton treated and the expressions for the cost of treating by both methods, selected from equation 9, we get:

$$(d) \quad \begin{aligned} & ABy - [(M + F_s + S_o) + Ay(R + E + c)] = \\ & ABCy - [(M + F_o + m) + \frac{(f_o + S_o)}{D}AC + (R + E + c)ACy] \end{aligned}$$

By substitution of the more fixed costs, etc., we get:

$$(e) \quad 1900AB - 8.8294 - 22.8A - 19AB = 1900ABC - 1.157 - \frac{7.4711AC}{D} - 22.8AC - 19ABC$$

Simplifying:

$$(f) \quad 1881AB - 22.8A = 1881ABC + 7.6724 - \frac{7.4711AC}{D} - 22.8AC$$

Factoring and transposing, we get:

$$1881AB(1 - C) = 7.6724 + 22.8A(1 - C) - \frac{7.4711AC}{D}$$

Transposing, we get:

$$1881AB(1 - C) - 22.8A(1 - C) = 7.6724 - \frac{7.4711AC}{D}$$

Multiplying through by  $-\frac{1}{A}$  and transposing, we have:

$$\frac{7.6724}{A} = 1881B(1 - C) - 22.8(1 - C) + \frac{7.4711C}{D}$$

Factoring and solving for A, we get:

$$\frac{7.6724}{A} = (1881B - 22.8)(1 - C) + \frac{7.4711C}{D}$$

Therefore:

$$(g) \quad A = \frac{7.6724}{(1881B - 22.8)(1 - C) + \frac{7.4711C}{D}}$$

Now, by substituting values for B, C, and D, we can solve for A. The value of A thus obtained will be the critical grade or percentage of copper per dry ton of ore, at which the profits will be equal whether the ore be smelted in the crude or milled and the concentrates smelted. If we let D = 20 per cent, C = 60 per cent, and B = 12 cents, we can substitute these values in formula *g* and solve for A, thus:

$$(h) \quad A = \frac{7.6724}{(1881 \times 0.12 - 22.8)(1 - 0.60) + \frac{7.4711 \times 0.60}{0.20}} = 0.0741, \text{ or } 7.41 \text{ per cent.}$$

If the copper content be greater than 7.41 per cent under the conditions given, the ore should be sent direct to the smelter; if less, it should be sent to the mill.

From the formula represented by equation *h* a table may be made showing the values of *A* for several values of *B*, *C*, and *D*, thus:

*Values of A for several values of B, C, and D, in equation h.*

D=20 per cent; C=60 per cent.		D=20 per cent; C=65 per cent.		D=20 per cent; C=70 per cent.	
B	A	B	A	B	A
12 cents.....	7.41 per cent	12 cents.....	8.05 per cent	12 cents.....	8.82 per cent
14 cents.....	6.47 per cent	14 cents.....	7.07 per cent	14 cents.....	7.80 per cent

By giving other values to *D*, as well as by giving *B* and *C* a large range, the table can be made much more extensive. In all cases the value of *A* thus obtained is the approximate grade of ore at which profits, per dry tone of ore, equalize, whether the treatment is direct smelting or milling and smelting the concentrates.

The expression  $\frac{M + F_s + S_o}{A_y} + R + E + c$ , taken from formula 8 (p. 30), represents the cost per pound of the copper sold when the ore is treated by direct smelting. By substituting values as given on page 30, simplifying, and rearranging, we get:

$$\frac{0.004647}{A} + 0.012 + \frac{B}{100} = 0.012 + \frac{0.004647}{A} + \frac{B}{100} = \text{cost per pound of copper sold when the ore is smelted direct.}$$

Now, by substituting values for *A* and *B*, and solving, we get the cost of copper per pound. Subtracting this figure from the selling price gives the profit per pound. If considerable range be given to the values of *A* and *B*, a table may be made somewhat as follows:

*Cost and profit per pound of copper paid for when ore is smelted in the crude.*

Price of copper per pound.		Per cent of copper in the ore.						
		4	5	6	7	8	9	10
<i>Cents.</i>								
12.....	{Cost.....	\$0.129	\$0.1061	\$0.0907	\$0.0796	\$0.0713	\$0.0648	\$0.0597
	{Profit....	.009	.0139	.0293	.0404	.0487	.0552	.0603
14.....	{Cost.....	.1296	.1063	.0909	.0798	.0715	.0650	.0599
	{Profit....	.0104	.0337	.0491	.0602	.0685	.0750	.0801
16.....	{Cost.....	.1298	.1065	.0911	.0800	.0717	.0652	.0601
	{Profit....	.0302	.0535	.0689	.0800	.0883	.0948	.0999
18.....	{Cost.....	.1300	.1067	.0913	.0801	.0719	.0654	.0603
	{Profit....	.0500	.0733	.0887	.0999	.1081	.1146	.1197
20.....	{Cost.....	.1302	.1069	.0915	.0803	.0721	.0656	.0605
	{Profit....	.0698	.0931	.1085	.1197	.1279	.1344	.1395

<sup>a</sup> Loss of 9 mills per pound of copper.

By substituting 0.05 for *A*, or 5 per cent, and 0.12 for *B*, or 12 cents per pound, and solving, we get \$0.1061 as the cost per pound of copper by direct smelting of 5 per cent ore, as shown in the above table; subtracting this value from the price of copper, \$0.12, we get \$0.0139 as the profit per pound. An extensive table may be compiled in this way.

In like manner, from formula *d* (p. 32) we get the expression  $[M + F_s + S_o + Ay(R + E + c)]$ , which stands for the total cost of mining and treating each dry ton of ore when the process is direct smelting. By substitution of the values given on page 30 the expression reduces to  $8.8294 + 22.8A + 19AB$ , the cost per dry ton of ore treated. Substituting values and solving gives the cost per ton. By subtracting the figure for cost from the product of the expression  $2000 \times 0.95 \times 0.05 \times 0.12$ , which represents the net pounds of copper recovered from each ton of 5 per cent ore smelted, multiplied by its selling price, we get the profit per ton treated. A table as extensive as may be necessary can be made thus:

*Cost and profit per dry ton (2,000 pounds) of ore smelted in the crude.*

Price of copper per pound.		Per cent of copper in the ore.						
		4	5	6	7	8	9	10
<i>Cents.</i>								
12.....	(Cost.....	\$9.833	\$10.083	\$10.334	\$10.585	\$10.836	\$11.087	\$11.337
	(Profit.....	a. 713	1.317	3.346	5.375	7.402	9.433	11.463
14.....	(Cost.....	9.848	10.102	10.357	10.612	10.866	11.121	11.375
	(Profit.....	.792	3.198	5.603	8.008	10.414	12.819	15.225
16.....	(Cost.....	9.863	10.121	10.380	10.638	10.897	11.155	11.413
	(Profit.....	2.297	5.079	7.860	10.642	13.425	16.205	18.987
18.....	(Cost.....	9.878	10.140	10.403	10.665	10.927	11.189	11.451
	(Profit.....	3.802	6.960	10.117	13.264	16.438	19.591	22.750
20.....	(Cost.....	9.893	10.159	10.426	10.692	10.957	11.223	11.489
	(Profit....	5.307	8.841	12.374	15.897	19.450	22.977	26.512

<sup>a</sup> Represents a loss of 71.3 cents per ton of ore treated.

By giving a larger range of values to A and B, this table can be greatly extended.

From formula 8 (p. 30) we get the expression  $\frac{(M + F_o + m)}{ACy} + \frac{(f_o + S_o)}{Dy} + (R + E + c)$ , which represents the cost per pound of the copper actually marketed where the ore is milled and the concentrates smelted. Now, substituting cost values given on page 30 in this expression we get  $\frac{1.157}{1900 AC} + \frac{7.4711}{1900 D} + 0.012 + \frac{B}{100}$  = cost per pound of copper sold when the treatment is milling and smelting the concentrates. By giving A, B, C, and D different values a whole series of tables may be compiled showing the cost and the profit per pound of copper sold.

In like manner, from formula *d* (p. 32), the expression  $[M + F_o + m + (\frac{f_o + S_o}{D})AC + (R + E + c)ACy]$  represents the cost of treating each dry ton of ore by milling and smelting the concentrates. Substituting costs gives  $1.157 + \frac{7.4711AC}{D} + 22.8AC + 19ABC$ . By substituting a range of values for A, B, C, and D an extensive series of tables can be compiled showing the cost and profit on each dry ton of ore by this method of treatment.



## SAMPLING AND ASSAYING METHODS.

## PROSPECT-HOLE SAMPLES.

In the development of "ore in sight" by drilling, samples were taken about every 5 feet of hole drilled in the relatively barren material near the surface; but in all other material they were taken every 3 feet. The mixture of water and drill-hole cuttings that composed each sample was run through what was, in effect, a Jones sampler, and quartered down so that sample and washings constituted about a gallon. This ultimate sample was evaporated to dryness on a specially arranged home-made furnace, consisting of a fire box and

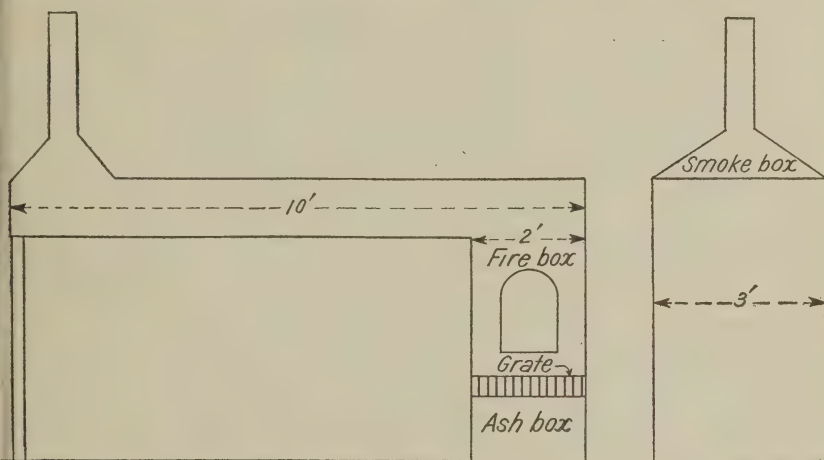


FIGURE 4.—Side and end views of improvised furnace for evaporating drill samples of ore. Can be made by blacksmith out of  $\frac{3}{8}$ -inch or  $\frac{1}{2}$ -inch boiler plate. Smoke box of heavy sheet iron; smokestack, 6 feet of ordinary 6-inch stovepipe; evaporating pans, about 18 by 18 inches and 6 inches high, made of heavy sheet iron.

a heating chamber built from  $\frac{3}{8}$ -inch boiler plate, somewhat as shown in figure 4.

Some of the rejected part of each sample was carefully panned and the concentrates examined to get information regarding the character of the rock through which the hole was passing. When notes covering these data had been made, the panned material was put into paper envelopes and filed away.

The dried sample was tagged with the number of the hole and the depth at which it was taken, and sent to the sample room. There, if necessary, it was run through a small gyratory crusher, the largest pieces from which were about one-fourth of an inch in diameter; then after careful rolling on a mixing cloth it was reduced on a Jones sampler to about 150 or 200 grams. After that came grinding in a Braun disk pulverizer, followed by more rolling on a mixing sheet. Screening through a 150-mesh to 200-mesh screen removed

any metallic copper, and this, if present, was weighed and figured into the ore values as 90 per cent pure. The pulp was then sent to the assay shop, where the determination was made, mostly by Lowe's potassium permanganate method. Each assay was given a serial number in addition to the number of the hole and the depth below the surface from which it had been taken, and the part of the pulp not required in the assay was put into a paper envelope and carefully filed away.

#### GOLD ASSAYS.

Since the summer of 1914 a large number of gold assays have been made. As a rule, the amount of gold present is small, so that charges of 4 assay tons <sup>a</sup> have had to be used to get a gold bead of a size that might be weighed conveniently. The smelter pays for the gold in the concentrates when its value is 60 cents or more per ton. Assuming a concentration of 10 into 1, and a saving of 70 per cent in the mill, a gold content of only 9 cents per ton in the ore would give a revenue from the smelter of 63 cents per ton of concentrates. Obviously the presence of even small amounts of gold favors the practice of making the concentrates as clean as possible, and during the winter of 1914-15 important progress was made in this direction at the Hurley mill.

#### BLAST-HOLE AND PIT SAMPLES.

Where more detailed information is necessary as mining proceeds, samples are taken from the blast holes. Such sampling shows with considerable exactness the value of the rock in place ahead of the steam shovels. Samples are taken—a small shovelful here and there—from the broken rock in front of each steam shovel. These are called pit samples and may be gathered every hour or so. Rough determinations, by the colorimetric method, are made on these, as they are only for the purpose of showing whether the rock is running about as it has been indicated by the cross sections and the estimates. Samples are also taken from every car of ore loaded, and these are checked against other assays of the ore body and against mill returns, tailings assays, etc. Formerly, samples were also taken by carefully cutting across the face of many of the benches with a small pick or a hammer and moil; a ladder was used to get at the upper part of the face. Few bench samples are taken now. Moisture determinations were made at various times on the ore and rock.

#### STEAM-SHOVEL SAMPLES.

Steam-shovel samples are occasionally taken and consist of a "grab" shovelful from each steam-shovel dipperful after it has been put on the car.

---

<sup>a</sup> An assay ton contains 29,166 grams.

To summarize, all four kinds of samples are currently taken at Santa Rita, as follows:

1. Prospect-hole samples.
2. Pit samples.
3. Car and steam-shovel samples.
4. Blast-hole samples.

Moisture determinations are made on many of the samples. On the average, about 2,400 assays and determinations are made per month. The assay staff consists of an assayer, one assistant, and two men in the sample shop. The latter attend to the grinding of the samples, preparing the pulp, etc.

#### SAMPLE SHOP AND ASSAY SHOP.

The sample shop, a one-story structure, having a floor space of about 500 square feet, is fitted with the following appliances: 1 Blake jaw crusher, with opening 10 by 5 inches, crushing to 2 inches, to which a revolving sampler is attached; a small Blake crusher, with an opening 3 by 4 inches, crushing to  $\frac{1}{4}$  inch; a gyratory crusher giving a maximum size of  $\frac{1}{4}$  inch; a Braun disk pulverizer; a gasoline stove for drying, and many other accessories. Power is furnished by an Allis-Chalmers 7.5 horsepower motor.

The assay shop, with almost 1,000 square feet of floor space, contains a balance room, a laboratory for wet assays, and a furnace room. In the latter are a smelting and a cupelling furnace, each fired by gasoline, vaporized under about 50 pounds of air pressure, and blown in through Cary hydrocarbon burners.

#### NUMBER AND COST OF ASSAYS.

The following table gives some idea of the number of assays made per month and the cost of making them:

##### *Cost of sampling and assaying at Santa Rita.*

Month.	Number of assays.	Cost per assay.	Mining costs.			Stripping costs.		
			Labor.	Supplies.	Cost per ton.	Labor.	Supplies.	Cost per cubic yd.
1914.								
January.....	3,610	\$0.46	\$610.73	\$106.69	\$0.00366	\$802.72	\$128.39	\$0.00302
February.....	2,159	.80	575.72	112.69	.00365	882.56	149.88	.00365
March.....	3,001	.66	623.36	151.46	.00349	997.96	193.41	.00345
August.....	a 1,335	.86	370.84	182.67	.00454	408.74	182.38	.00317
September.....	b 1,475	.42	207.90	58.03	.00255	276.14	78.74	.00223

<sup>a</sup> 66 of these assays were for gold.

<sup>b</sup> 130 of these assays were for gold.



As is indicated by the table, the figures cover ore and waste. The ore assay costs are charged against mining, and the costs of assaying waste are charged against stripping. The cost of the labor for assaying and sampling and the cost of the supplies used in connection with the assays are kept separate, as they are in all the other accounts.

The assay cost is distributed per ton according to the tons of ore mined, and per cubic yard according to the cubic yards of waste moved, as shown by the table.

The permanganate method is used for prospect drill holes and for car samples. The colorimetric method is used for pit and shovel samples where absolute accuracy is not considered necessary. The determination of iron in many samples is made as it is indicative of the degree to which the ore will concentrate.

The wide variation in the cost of the assays is caused by the wide variation in the amount of crushing and mixing to be done on them—whether permanganate or colorimetric determinations or fire assays are made; the relative number of each, and the total number of all assays for the month; the amount and cost of the chemicals used, etc.

#### FORMS USED IN ASSAY SHOP.

Forms 6 to 9 following are used in the assay shop and are convenient aids to bookkeeping.

FORM 6.—*Form of general assay certificate issued by assayer.<sup>a</sup>*

#### ASSAY CERTIFICATE.

Per ton of 2,000 lbs.

SANTA RITA, N. MEX., ..... 191...

Sample No.	Date.	S. S. No.	Ore body.	Bench.	Description.	% Silica.	% Sulphur.	% Iron.	% Copper.	Oz. gold.	Oz. silver.				

..... Assayer.

<sup>a</sup> Size of form as used by operating company, 5½ by 7½ inches.

FORM 7.—Form used by assayer to report on steam-shovel samples.<sup>a</sup>

## STEAM-SHOVEL SAMPLES—LABORATORY REPORT.

Sample No.	S. S. No.	Date.	Shift.	Cars waste.	Cars ore.	Wt. of met.	Wt. of pulp.	% Met. Cu.	% Perm. Cu.	% Cu. total.	

<sup>a</sup> Samples consisted of a small shovelful taken from each steam shovelful that went onto a car. Samples of this kind are not now usually taken. Form used by operating company is 6 inches wide by 9 inches long.

FORM 8.—Form for presenting comparative data regarding steam-shovel samples.<sup>a</sup>

## COMPARATIVE STATEMENT—STEAM-SHOVEL SAMPLES.

DAY SHIFT....., 191..

Steam Shovel No.....

..... Ore body.

..... Bench.

Time.	Pit sample % Cu.	Mined as—		Bench sample—		Mined as—	
		Ore.	Waste.	No.	% Cu.	Ore.	Waste.
Average % Cu.							
Car sample % Cu.							
Car sample % Cu. Ore shipped to mill.							

## SPECIAL PIT SAMPLES.

Time.	Taken from.	Ore % Cu.	Waste % Cu.
Remarks:			

<sup>a</sup> Only one form is shown; the record used by the operating company has five of these forms (3½ by 8½ inches) on a sheet, the sheet with margins being 10 by 20½ inches. Each sheet, bound through two holes in the left margin, is kept in a heavy binder.

FORM 9.—*Form used in making daily report on mine samples.<sup>a</sup>*

## DAILY REPORT OF MINE SAMPLES.

.....19....

Mine sample number.	Description.			Per cent copper.			Remarks.
	Mine.	Level.	Working.	Pulp.	Met.	Total.	

<sup>a</sup> Size of form used by operating company, 8½ inches wide by 10¼ inches long.

## METHOD OF SAMPLING OLD DUMPS.

Sampling on the present property, on an extensive scale, was first begun by Mr. John M. Sully, in 1906. In gathering samples from the large dumps left from many thousands of feet of old workings, he used the following method:

The dumps were laid off into 25-foot squares, with levels run to the corners of each square, to determine its elevation. The contour of the ground on which the dump rested was estimated as nearly as possible, from the general surface configuration of the land. From the surface elevation and the probable elevation of the bottom of the dump, the yardage in each square prism was estimated. Each square, as laid off on the dump surface, was given a number, and about 4 holes were dug into it, as deep as could be conveniently made with a long-handled shovel. Every second shovelful was retained; the sample thus gathered was rolled and quartered on a canvas and put into a tagged canvas sack. Mr. Sully gathered 1,143 samples from the old dumps, and a total of 8,190 from the dumps and the old workings.

## DETAILS OF MINING IN SANTA RITA DISTRICT.

## GENERAL ORGANIZATION OF OPERATING COMPANY.

In opening an open-cut mine on an extensive scale, the initial cost is heavy. The equipment necessary is expensive, including steam shovels, locomotives, and cars, and railroad track and a water-supply system must be constructed, to say nothing of the concentrating plant to handle vast tonnages of ore, the power plant to furnish the motive force, and the machine and other shops to keep the whole vast plant in repair.



Any large and successful organization is developed slowly. The factor that wins in business or in sport is practically always well-organized "team work," or efficiency of production. If a mistake occurs the organization must be such that the responsibility for the mistake can be traced to its source at once, and the right remedy be quickly applied in the right place. The management of the Chino company has been successful in building up an efficient and loyal organization, one illustrating the benefits that may be derived from the application of science and of business methods to the solution of mining problems. The company's organization chart, or "flow sheet" of responsibility, as worked out by Mr. L. E. Foster, the assistant superintendent, is shown in Plate IV.

#### MINING AND ESTIMATION OF ORE BY BENCHES.

In stripping and mining, each steam shovel took off a slice of ground about 25 feet wide, and as the work progressed terraces or benches 100 feet or more wide were formed around the sides of the great open cuts (Pl. V, A). It was found that where the bench was less than 30 feet high, the material could not be excavated as economically as where it was, say, 50 feet or more high.

There were three chief factors that regulated the height of each bench: (1) If a bank stood fairly steep for some time and then caved, it was found best to have the bench not more than 30 feet high. The caving of 30-foot banks caused little delay or damage to the shovels working below them, whereas the caving of higher banks often caused serious trouble. Where a bank caved slowly and kept gradually coming to the shovel at about the same general slope, it could be worked in benches of 100 feet or more if necessary. (2) If the waste material on top of the ore was, say, only 15 feet thick, then, of course, the bench at that place was only 15 feet high. On the other hand, if the overburden was only, say, 1 or 2 feet thick, it was probably found cheaper to let it go in with the ore without stripping. (3) Where the drill holes indicated considerable relatively barren rock between upper and lower ore masses, then, as far as possible, the benches were planned so that the top ore was taken off the waste rock in one cut or bench, and the waste rock off the lower ore in the second bench, and the lower ore was taken out clean in a third bench. If the ore bodies and waste mass were thick enough, they were generally taken out in two or more separate benches. Barren areas of considerable size were also considered in planning the benches.

In other words, benches and tracks were planned (see Pl. III) to conform, as nearly as practicable, to the shape of the ore body, so that ore could be separated from waste and mined as economically as possible. In general, however, it was found best to have the benches about 50 feet apart; that is, one bench about 50 feet above the other.

The different classes of ore and the waste were calculated for each level, or bench, in order to have convenient information as to just what production should be expected from each.

Tables were also made to show the ore tonnage by benches; that is, the tonnage of the first bench plus the tonnage of the third bench, etc. Such tables permit convenient comparison of what has been taken out from the different benches plus the material calculated to remain with the total ore for each bench, etc. These bench calculations were made from the same data and by the same general method as the triangular-prism estimates. Summing up the ore classes, calculated by benches, served as a sort of check on the other calculations of total ore.

#### BLAST-HOLE DRILLING.

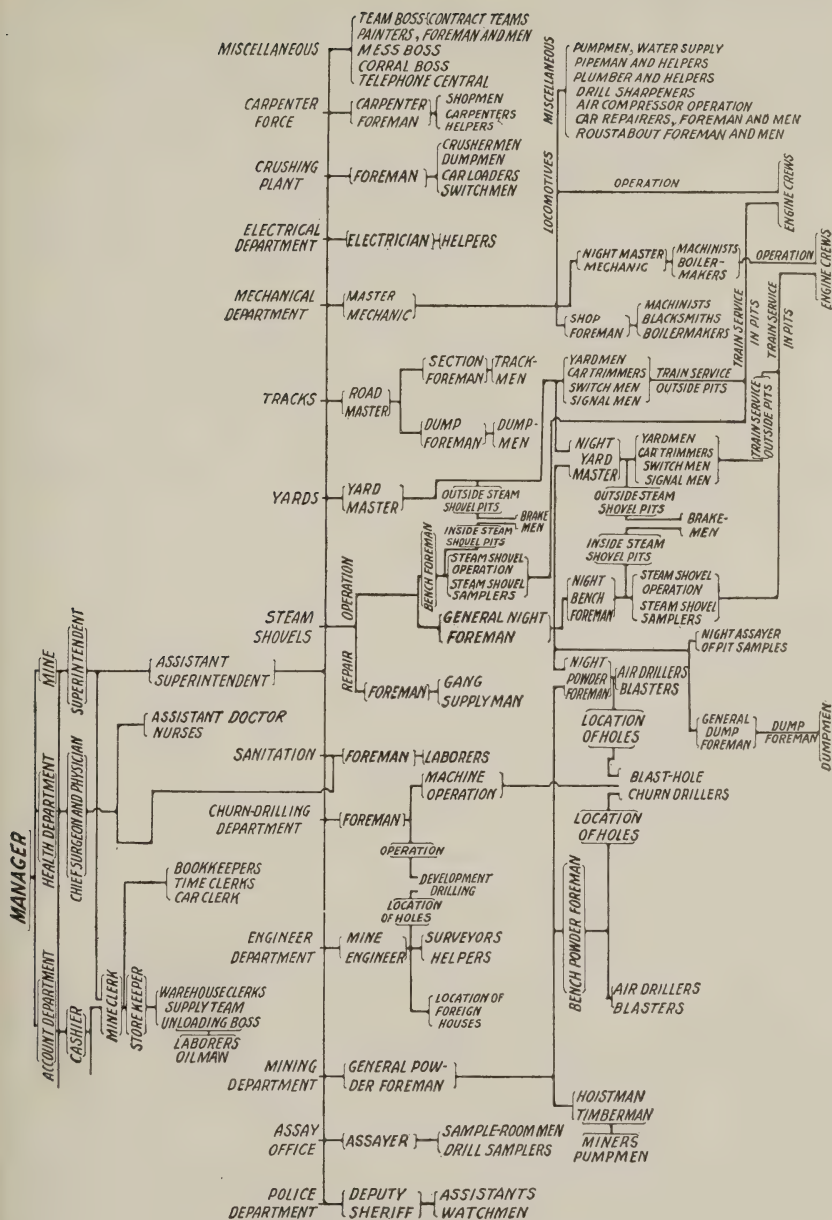
The blast-hole drilling at Santa Rita aggregates several linear miles per month. Most of it is done by churn drills. However, toe holes at the foot of slopes are locally necessary, and these, together with the shallow, levelling-off bottom holes occasionally needed in front of steam shovels, are put down with compressed-air tripod drills. Small hammer drills have been used, to some extent, for drilling short holes to blast boulders too large for the steam shovels to handle. However, their use has been practically discontinued, because it is generally cheaper to use more powder and to "dobe" or "bulldoze" the rock than to keep a shovel idle while it is being drilled.

#### TYPES OF CHURN DRILLS USED.

The types of churn drill mostly used at Santa Rita are the No. 12 and the No. 14 Cyclone drills for big blast holes. They are fitted with Cook 12-horsepower vertical gasoline motors, have traction gearing, and use 4-inch to 8-inch bits. Gas-engine oil, instead of gasoline, is used, and the consumption averages about  $6\frac{1}{2}$  gallons per 10-hour shift. The speed of drilling varies widely with variation in the hardness of the rock; however, the general average for blast holes not more than 65 feet deep is said to be about 20 to 30 feet per 10-hour shift. The crew of each machine consists of a driller and a helper; these two men can move the drill, even over rough ground, by turning on the traction gearing.

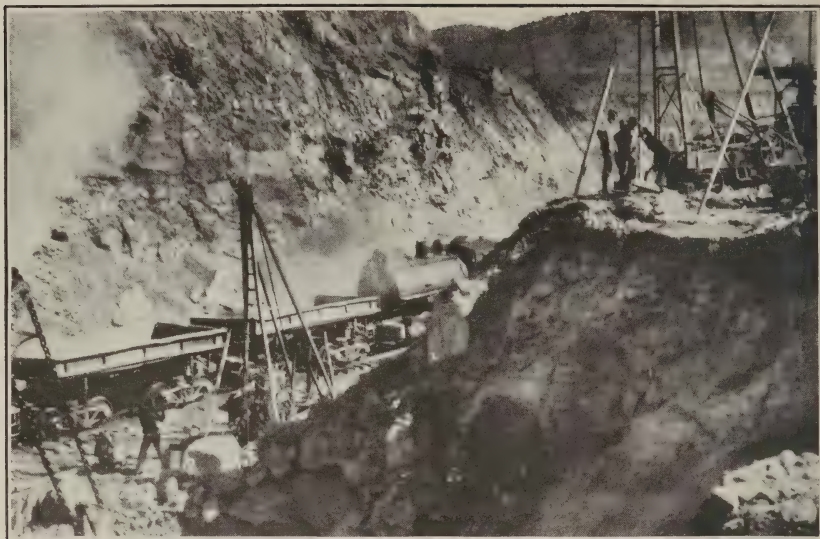
Each drill is fitted with equipment that enables the driller to sharpen his own bits. In general, the rock is soft enough so that one to two sharpenings per shift suffice. The blast holes are sampled if special information from particular localities is needed.

A few of the older steam-driven churn drills are now operated by air, piped from the compressor. In general, they are not as satisfactory as the oil-motor machines. There have been as many as 18 churn drills, both prospect and blast-hole, at work on the property at the same time.



ORGANIZATION CHART OF THE CHINO COPPER CO.





A. VIEW OF A BENCH OR TERRACE OF ORE, SHOWING METHOD OF MINING.

On the right, drills are at work on a bench just ahead of a steam shovel. The chain of the shovel can be seen at the left. A train of cars is on the loading track below the bench that is being drilled.



B. PILE OF COAL CONTAINING WOODEN VENTILATING STACKS TO PREVENT SPONTANEOUS COMBUSTION.

The hydrothermally altered and considerably sheared character of the ore-containing rock facilitates the use of churn drills and is favorable for low drilling costs. Little trouble is encountered in putting down blast holes. Even in cutting across sloping sheared zones, a little care generally guards against much deviation from the vertical.

## OPERATION AND EFFICIENCY OF BLAST-HOLE CHURN DRILLS.

Cyclone churn drills, models Nos. 12 and 14 for deep blast holes, are used to put down blast holes. Records of the performance of these are kept and constitute valuable information on the efficiency of churn drilling in general. Records for two typical months, January and August, 1914, are given below:

*Record of blast-hole churn drilling for January, 1914.*

## DAY SHIFT.

Drill No.	Feet drilled.	Total working hours.	Average feet per hour.	Time spent in drilling.	Time delayed.	Time spent in repairing.	Average feet per hour of drilling.
				<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
1.....	630	280	2.25	72.8	19.3	7.9	3.09
2.....	876	310	2.83	61.9	25.2	12.9	4.56
10.....	1,235	327	3.78	84.1	11.9	4.0	4.49
11.....	974	310	3.14	75.2	16.1	8.7	4.13
12.....	1,212	310	3.91	86.8	8.4	4.8	4.51
13.....	941	322	2.92	73.9	15.8	10.3	3.95
14.....	924	311	2.97	78.5	17.0	4.5	3.79
15.....	952	310	3.07	79.0	20.0	1.0	3.88

## NIGHT SHIFT.

1.....	670	260	2.58	86.5	13.1	0.4	2.98
2.....	1,144	298	3.84	84.2	13.8	2.0	4.56
10.....	1,272	321	3.95	88.2	7.8	4.0	4.50
11.....	928	266	3.49	89.9	8.6	1.5	3.88
12.....	1,244	290	4.29	98.0	1.7	.3	4.38
13.....	1,000	329	3.04	83.6	15.8	.6	3.64
14.....	953	275	3.47	90.5	9.5	.....	3.83
15.....	1,151	310	3.71	90.3	9.4	.3	4.12

## TOTAL.

1.....	1,300	540	2.41	79.4	16.3	4.3	3.03
2.....	2,020	608	3.32	72.8	19.6	7.6	4.56
10.....	2,507	648	3.87	86.1	9.9	4.0	4.49
11.....	1,902	576	3.30	81.9	12.7	5.4	4.03
12.....	2,456	600	4.09	92.2	5.1	2.7	4.44
13.....	1,941	651	2.98	78.8	15.8	5.4	3.78
14.....	1,877	586	3.20	84.1	13.5	2.4	3.81
15.....	2,103	620	3.39	84.7	14.7	.6	4.01
Total for month.....	16,106	4,829	3.34	82.5	13.4	4.1	4.04

*Record of blast-hole churn drilling for August, 1914.*

## DAY SHIFT.

Drill No.	Feet drilled.	Total working hours.	Average feet per hour.	Time spent in drilling.	Time delayed.	Time spent in repairing.	Average feet per hour of drilling.
				<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
1.....	725	186	3.90	95.2	4.8	.....	4.09
2.....	1,073	235	4.57	95.3	4.7	.....	4.79
10.....	1,049	244	4.30	91.8	2.0	6.2	4.68
11.....	1,082	256	4.23	92.6	4.7	2.7	4.56
12.....	788	187	4.22	89.8	7.0	3.2	4.69
13.....	814	232	3.51	83.2	2.1	14.7	4.22
14.....	383	185	2.07	75.1	18.4	6.5	2.75
15.....	963	246	3.91	86.6	8.5	4.9	4.52
16.....	988	246	4.02	94.3	2.0	3.7	4.26

## NIGHT SHIFT.

Drill No.	Feet drilled.	Total working hours.	Average feet per hour.	Time spent in drilling.	Time delayed.	Time spent in repairing.	Average feet per hour of drilling.
1.....	.....	.....	.....	.....	.....	.....	.....
2.....	125	30	4.17	100.0	.....	.....	4.17
10.....	477	129	3.76	97.7	0.8	1.5	3.79
11.....	82	39	2.10	74.3	23.1	2.6	2.83
12.....	182	44	4.14	91.0	4.5	4.5	4.56
13.....	304	81	3.75	95.1	4.9	.....	3.95
14.....	75	39	1.92	69.2	28.2	2.6	2.78
15.....	.....	.....	.....	.....	.....	.....	.....
16.....	352	94	3.74	95.7	4.3	.....	3.92

## TOTAL.

Drill No.	Feet drilled.	Total working hours.	Average feet per hour.	Time spent in drilling.	Time delayed.	Time spent in repairing.	Average feet per hour of drilling.
1.....	725	186	3.90	95.2	4.8	.....	4.09
2.....	1,198	265	4.52	95.8	4.2	.....	4.72
10.....	1,526	373	4.09	93.8	1.6	4.6	4.36
11.....	1,164	295	3.95	90.1	7.2	2.7	4.38
12.....	970	231	4.20	90.0	6.5	3.5	4.67
13.....	1,118	313	3.57	86.2	2.9	10.9	4.14
14.....	458	224	2.04	74.0	20.2	5.8	2.76
15.....	963	246	3.91	86.6	8.5	4.9	4.52
16.....	1,340	340	3.94	94.6	2.7	2.7	4.16
Total for month.....	9,489	2,473	3.84	90.0	5.9	4.1	4.26

## COSTS OF BLAST-HOLE CHURN DRILLING.

The cost of drilling blast holes with churn drills are fairly well illustrated by the figures for the representative months of January and August, 1914, as given in the tabulation following:

*Record of costs of blast-hole churn drilling.*

AUGUST, 1914.

Churn drill No.	Cost of labor.	Cost of supplies.	Total.	Feet drilled.	Cost per foot.
1.....	\$229.43	\$73.36	\$302.79	725	\$0.4176
2.....	311.52	76.49	388.01	1,198	.3239
Total for steam drills using air.....	540.95	149.85	690.80	1,923	.3592
10.....	383.42	58.25	441.67	1,526	.2894
11.....	317.99	100.68	418.67	1,164	.3597
12.....	302.49	89.93	392.42	970	.4046
13.....	361.37	147.90	509.27	1,118	.4555
14.....	274.08	106.00	380.08	458	.8299
15.....	261.02	34.17	295.19	963	.3065
16.....	377.13	70.19	447.32	1,340	.3338
Total for gasoline drills.....	2,277.50	607.12	2,884.62	7,539	.3826
Total for all churn drills.....	2,818.45	756.97	3,575.42	9,462	.3779



*Record of costs of blast-hole churn drilling—Continued.*

JANUARY, 1914.

Churn drill No.	Cost of labor.	Cost of supplies.	Total.	Feet drilled.	Cost per foot.
1.....	\$595.59	\$183.40	\$778.99	1,300	\$0.599
2.....	637.90	193.08	830.98	2,020	.411
Total for steam drills using air.....	1,233.49	376.48	1,609.97	3,320	.485
10.....	668.24	195.21	863.45	2,507	.344
11.....	594.89	242.66	837.55	1,902	.440
12.....	622.02	176.92	798.94	2,456	.325
13.....	653.24	190.99	844.23	1,941	.435
14.....	607.87	135.12	742.99	1,877	.396
15.....	618.97	244.88	863.85	2,103	.411
Total for gasoline drills.....	3,765.23	1,185.78	4,951.01	12,786	.387
Total for all churn drills.....	4,998.72	1,562.26	6,560.98	16,106	.407

The total cost of drilling per ton of ore mined is about 1 cent; and per cubic yard of material stripped, about 1.3 cents. The reasons for the cost per foot of blast-hole drilling being so much less than the cost of prospect drilling (p. 14) are that the prospect holes were larger and averaged eight to ten times as deep as those for blasting; also the cost of sampling was included in the prospect-hole costs.

## TYPES OF TRIPOD DRILLS USED.

Two types of tripod drill, the Sullivan and the Ingersoll-Rand, are used where the rock is hard enough so that toe and shallow bottom holes are necessary. Both these makes, having 3¼-inch and 3⅝-inch cylinders, were satisfactory. They are operated by air under about a 90-pound pressure, piped from the central compressor plant. The steel used is mostly of the 1¼-inch crescent brand. It is sharpened by a No. 3 Leyner drill-sharpening machine. Machine-sharpened drills have given better service at a smaller cost than was obtained from hand-sharpened drills.

## COSTS OF DRILLING BLAST HOLES BY AIR DRILLS.

During January, 1914, air-driven tripod drills drilled 9,875 linear feet of holes at a cost of 46½ cents per foot; during August they made 6,951 feet of holes at the rate of 32½ cents per foot. This great difference in rate of cost seems to be due to greater ease of drilling during August.

## USE OF HAMMER DRILLS. •

Where boulders, too large to be handled by a steam shovel, remain after a bench blast, and where loading will not be too much impeded by drilling, or "block holing," instead of by "dobe" blasting, the "block holing" is done by small air hammer drills of the Sullivan D. C., 19 type, using 2-inch bits. Air is supplied to them through flexible armored hose from a supply line running along the top of

the cuts. The exhaust from the machine passes out through a hole in the bit, thereby automatically cleaning the hole of cuttings. From  $1\frac{1}{2}$  to 3 inches of hole can be drilled per minute with this machine, and it can be operated by one man. The operator is paid \$3.75 per day. Only one of these drills was in use at the time the mine was visited, because it had been found that "dobe" blasting of boulders could be done more quickly and interfered less with loading operations, and hence, in the long run, was more economical.

#### BLASTING.

The object of blasting in metal mines where there are rich brittle sulphides and carbonates is generally, as in coal mines and quarries, to break material loose from the main mass without pulverizing it greatly. On the other hand, if the broken rock is subsequently to be crushed for recovery of its values or for engineering operations the aim then is that the blasting shall break the rock loose and at the same time pulverize it as much as is possible, consistent with the efficient use of explosives. At Santa Rita, of course, the aim in blasting the ore-bearing rock is to pulverize it. Any fragments of broken rock that remain too large to enter a 3-yard steam-shovel dipper have to be broken by secondary blasting at considerable extra cost.

#### EXPLOSIVES USED.

In breaking the rock in a safe and efficient manner the following kinds of explosives are those most extensively used:

- (a) Dupont FF and FFF black blasting powder.
- (b) Dupont quarry powder.
- (c) Dupont Repauno low-freezing dynamite, 30, 40, and 60 per cent strengths.

- (d) Trojan powder No. 2 in bags, and 40 per cent strength, in sticks.

Black powder is used for breaking and loosening up soil, hardpan, or soft rock; and quarry powder, instead of the 40 per cent dynamite, is used for soft or fissured rock. Repauno 40 per cent strength low-freezing dynamite is used to spring holes for primers and to blast holes that contain water; it is also used for holes in medium hard rock, and for "tight holes" in broken formations. For "dobe" shots 40 per cent Repauno dynamite is used almost exclusively. Repauno 60 per cent strength dynamite is used only in the hardest rock formations. Trojan powder, packed in  $12\frac{1}{2}$ -pound waterproof paper bags, four bags to the box, is used most in medium hard or broken rock. This powder is understood by miners to be of equal strength to 40 per cent dynamite, but on account of its loose, granulated form it fills in the cracks and crevices in broken formations, thus excluding air spaces and increasing the blasting efficiency. Trojan sticks are also used for "springing" or chambering wet holes.

Detonation by fuse and detonator is used for springing holes and for "dobe" or "bulldoze" blasting. All the heavy bank-hole blasting and some of the heavy springing charges are detonated with electric detonators. The brands of fuse used are "Sylvanite" and "Cactus." Sylvanite is an asphaltum-covered, triple-wound, cord fuse, which has a burning rate of 16 inches per minute. Cactus is a triple-wound cord fuse, with a white clay-compound waterproof covering. It has a burning rate of about 20 inches per minute, and on account of its white color is generally used for night work. No. 7 detonators are used with these fuses. Where the detonation is by electric current, No. 7 electric detonators are used.

#### STORAGE AND HANDLING OF EXPLOSIVES.

The main magazines have each a capacity of about 130 tons. Each is formed by a drift into the hillside so that a chamber about 30 by 25 feet, and 8 feet high, is cut into the solid rock (fig. 5). Each chamber is secured by double sets of timbers, with stulls or legs set about 5 feet apart, and with the roof and sides lagged. From the entrance to the back wall a 5-foot passageway is kept clear. The explosives are segregated according to kind and are piled in tiers in the chamber. These underground magazines afford safety against stray bullets, etc., and give what is necessary in the storing of explosives—a comparatively even temperature that remains above freezing during summer and winter. The floor of each of these chambers is covered with several inches of mill tailings. Thus a soft smooth surface, which acts as a cushion to the cases of explosives, is insured.

The magazines have to be kept well ventilated in order that they may not become overheated in summer and that gases from the stored materials may not accumulate in them. To this end, a 6-inch churn-drill hole connects the inner part of the passage in each magazine with the surface (fig. 5). These bore holes are protected on the surface with hooded stacks about 12 feet high, the upper 3 feet of which are perforated.

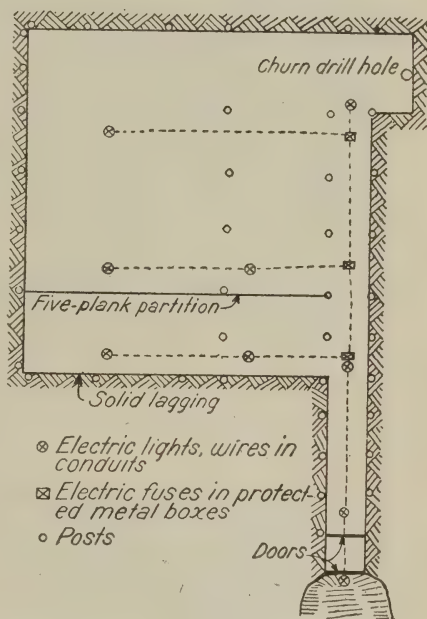


FIGURE 5.—Plan of typical underground explosives magazine in Santa Rita district.



The entrance to the magazines is guarded by two double-planked, steel-faced, swinging doors, one about 6 feet inside the other. The outside door is perforated in the upper half, and the inside one in the lower half, for the admission of air (Pl. VI, A).

From the magazines the explosives are distributed by wagons to several small portable powder stations. There are two of these stations to every steam shovel—one for the high explosives and one for black powder—so they are known as shovel powder houses. They consist of skeleton frames, made of  $2\frac{1}{2}$ -inch by  $2\frac{1}{2}$ -inch material, 7 by 4 feet by  $4\frac{1}{2}$  feet high, and covered with corrugated sheet iron. Each of these little houses has a capacity of 3,500 pounds of high explosives, or 4,000 pounds of black powder. However, they seldom contain more than a wagon-load—2,000 to 3,000 pounds. Carrying handles, consisting of scantlings 2 by 4 inches in size, are nailed to the sides.

When black powder is needed in large quantities at any point, a sled with a pointed corrugated sheet-iron roof and drawn by a team of horses, is used to transport it from the shovel powder house to the place of use.

Little trouble is experienced from the freezing of explosives, as the operating company generally buys only the low-freezing or nonfreezing brands, and as the magazine temperature is always above freezing. During exceptionally cold weather hot-water thawing chambers are used and then the "powder" has to be carried a considerable distance from the magazines.

#### "SPRINGING" OR CHAMBERING HOLES.

Before the drill holes are loaded and blasted, one or more small charges are exploded in the bottom of each, to make an enlarged chamber for the final charge. This is known as "springing" or chambering the holes. The aim is to make so large a chamber at the bottom of the hole that the charge may be concentrated into so compact a mass that none of it will fail to detonate, and that the explosive energy may be concentrated where it will give the maximum effect. The springing eliminates any water that may be in the hole, but it generally increases the liability of an inflow of water by shattering and fissuring the surrounding rock. If granulated powder can be forced into these fissured places the effectiveness of the blast is greatly increased.

Where the rock to be broken is relatively soft and weak, and the drill holes not more than, say, 30 feet deep and not over a third of that distance back from the face of a steam-shovel cut, one "springing" is usually all that is necessary. Where the rock is hard and tough, or where the holes are very deep, as many as four successive springing charges may be required before a chamber large enough for



4. ENTRANCE TO UNDERGROUND EXPLOSIVES MAGAZINE.



5. EXPLODING A SPRINGING CHARGE IN A BANK HOLE.

Note how the water and rock shoot out at the surface.





the final blasting charge is made. After each successive "springing" the hole must be cooled off, generally with water, before another charge is introduced.

The following are average charges for each successive "springing," in medium-hard rock:

*Average charges for "springing" or chambering drill holes.*

Springing No.	Bank holes.		Toe holes.	
	Number of primers used.	Number of sticks of dynamite used.	Number of primers used.	Number of sticks of dynamite used.
1.....	1	5	1	2
2.....	1	11	1	5
3.....	2	21	1	9
4.....	2	28		

For "springing" holes 40 per cent strength dynamite or other high explosive, cut into quarter or third sticks, is generally used. It is detonated with a No. 7 detonator crimped onto 24 inches of fuse.

Both before and after "springing" the drill holes are sounded by a specially constructed plummet—a billet of hardwood about 30 inches long and having a metal weight in the lower end. This sounding apparatus, known as a "springing gage" (fig. 6), is attached to a rope and lowered into the hole. By tilting the upper end of it from side to side while the lower end rests on the bottom (fig. 7) the experienced chargeman can estimate the size of the hole. With this gage he also measures the depth of water in the hole. A small hand mirror is at times very useful for reflecting

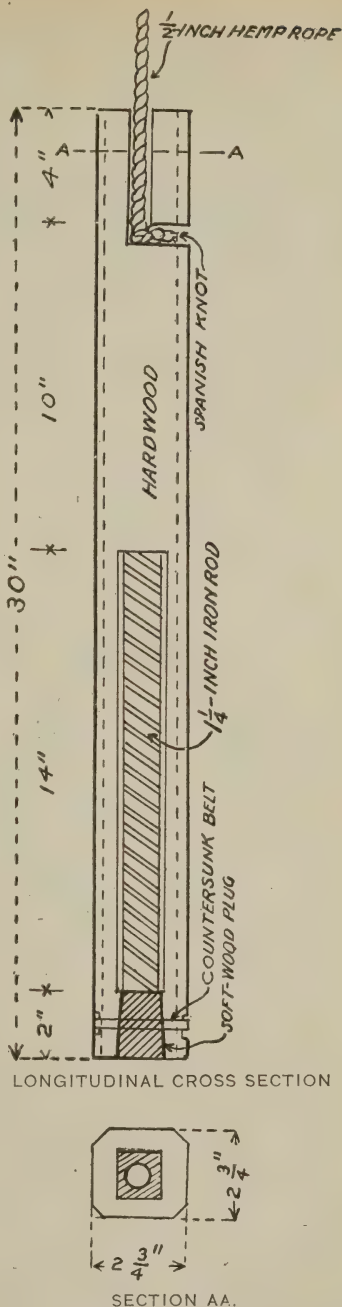


FIGURE 6.—Longitudinal and transverse cross sections of "springing gage."

light into the hole; thus enabling inspection of the upper 10 to 30 feet.

The procedure in "springing" holes is somewhat as follows: The chargeman lowers the springing gage into the hole and notes its size and depth, the depth of water, etc. He then drops in the springing charge, and again lowers the gage to determine whether the charge has reached the bottom. A fuse, passed diagonally through a stick of dynamite about 2 inches from one end, extends along the side of the stick to the other end, into which the end containing the detonator is inserted about  $2\frac{1}{2}$  inches (fig. 8, A).<sup>a</sup> This stick of dynamite, with cap and fuse attached, is called a "primer." The fuse is then lighted and the primer is dropped or lowered into the charged

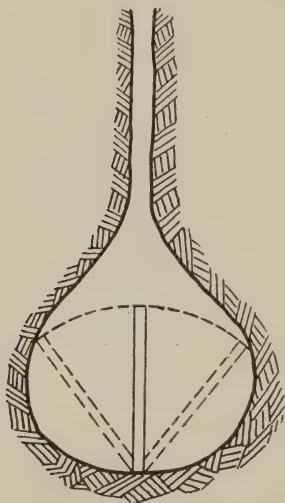


FIGURE 7.—View of chambered drill hole showing how "springing gage" is used.

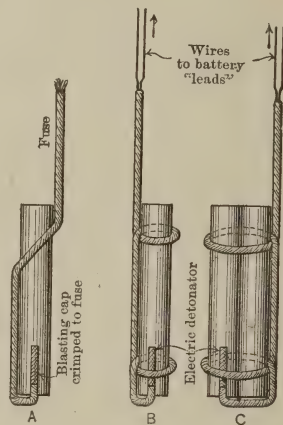


FIGURE 8.—Arrangement of fuses for different primers. A, for detonation with fuse, as practiced in Santa Rita district; B, for electrical detonation, single primer; C, for electrical detonation, double primer.

hole, followed by a hurried lowering of the gage to show whether the primer has reached the bottom. After this 2 to 4 gallons of water is poured in as "stemming," the gage is quickly withdrawn, and the chargeman retreats to a safe distance. After the detonation, the gage is again lowered and the size of the chamber estimated. If necessary other "springing" charges are used, each about twice as large as the preceding one, until the hole is sufficiently widened or chambered out at the base to receive a proper blasting charge (fig. 9). For the third "springing," where the holes are over 30 feet deep, and often for the second "springing," if much water is present, electric detonation is used (fig. 8, B and C). Where there are several holes to be fired simultaneously electricity is employed instead of fuse.

<sup>a</sup> The Bureau of Mines does not recommend the lacing of a fuse through a primer.

The lead wires or detonating wires are connected in series and the detonators tested with a galvanometer. The lead wires are then connected with a battery and the charges are fired (Pl. VI, B).

#### CHARGING AND FIRING.

The blasting squad usually consists of a charge man or powder man and four helpers. The charge man places all explosives in the drill holes, except in the "springing" of short holes, when the work may

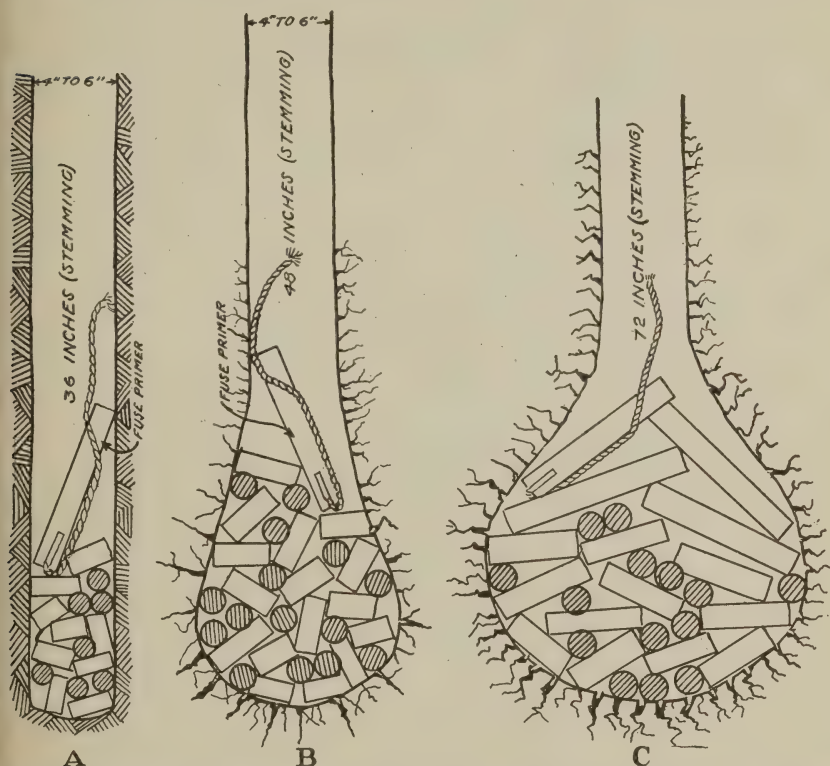


FIGURE 9.—"Springing" charges in bank holes in dry soil or soft rock. Water stemming only. A, charge for first "springing"; B, charge for second "springing"; C, charge for third "springing."

be done by the assistant charge man. The helpers carry the explosives and accessories to the charge man while he is gaging the hole and charging it.

The amount of the charge for each hole varies according to the toughness of the rock, the position of the hole with relation to the face and to other holes, and its depth and extent of chambering. The drill holes to be blasted are "bank holes"—those drilled vertically on each bench and situated back approximately one-third of their depth from the edge of the bench—and "toe holes"—those



driven into the face of the bench, say, 4 to 6 feet above the shovel level at the foot of the bench. The toe holes are driven with a downward slope of  $20^{\circ}$  to  $30^{\circ}$  below the horizontal.

Average charges for bank holes in medium-hard rock are about as follows:

*Average charges for bank holes in medium-hard rock.*

Depth of hole.	Sticks of dynamite in primer.	Charge.		
		40 per cent strength dynamite.	Trojan powder No. 2 (granulated).	Stemming.
<i>Feet.</i>		<i>Pounds.</i>	<i>Pounds.</i>	<i>Feet.</i>
28	1 and 2	138	100	24
40	1 and 2	.....	267	36
50	1 and 2	170	200	40
70	3	.....	550	60
<sup>a</sup> 50	2	<sup>b</sup> 375	.....	30

<sup>a</sup> Driven into soil.

<sup>b</sup> FF black powder.

Average toe-hole charges for medium-hard rock are:

*Average toe-hole charges for medium-hard rock.*

Depth of hole.	Number of sticks of dynamite in primer.	Charge.	
		40 per cent strength dynamite.	Stemming.
<i>Feet.</i>		<i>Pounds.</i>	<i>Feet.</i>
8	1	12.5	4 to 6
18	1 and 2	60.0	12 to 14

The following are average mixed charges from a number of observations:

*Average mixed charges.*

#### BANK HOLES.

Depth of holes.	Quantity of mixed charge.	Charge per foot of hole.
<i>Feet.</i>	<i>Pounds.</i>	<i>Pounds.</i>
28	188	6.72
40	267	6.68
50	369	7.38
<sup>a</sup> 50	<sup>b</sup> 375	7.50
70	550	7.86

#### TOE HOLES.

8	12.5	<sup>c</sup> 1.56
18	60.0	3.33

<sup>a</sup> In soil and soft rock.

<sup>b</sup> Black powder.

<sup>c</sup> Holes not chambered.

When the holes are sufficiently chambered or sprung and are ready for loading, an electric-detonator primer, prepared as shown in figure 8, C, is carefully lowered to the bottom of the hole. The first part of the charge, varying from 100 to 300 pounds, is dropped in after it. Another primer is then placed in the hole and the remainder of the charge, 75 to 250 pounds, is added (fig. 10). During the process of loading, the charge man frequently drops in his gage, to determine how rapidly the hole is filling and to guard against the clogging of the powder. Often granulated Trojan powder is poured in to fill the interstices between the sticks of dynamite and the fissured places in the rock (fig. 10). Thus air spaces are minimized and the efficiency of the blast greatly increased. When the charge is all placed, screened clayey sand and loam are carefully shoveled into the hole and packed down for stemming.

All blasting is done at such time as will least interfere with the operation of steam shovels, trains, drilling machines, and general work. Wherever practicable, the blasting is done at the end or before the beginning of the shift or during the noon period. A great deal of blasting is also done by the night squad.

The rules of the company prohibit more than one independent charge in any hole; that is, the explosives must be so placed that no stemming separates any part or parts of the total charge in any hole. Double or triple charges, separated by stemming, are not allowed to be put into one hole, owing to the danger from incomplete detonation. A part of the charge under such conditions might not explode, thus rendering subsequent steam-shovel operations dangerous, for unexploded dynamite is sometimes set off when dug into by steam shovels.

The toe holes are loaded in much the same way as the bank holes, except that the charge has to be pushed into the relatively flat toe

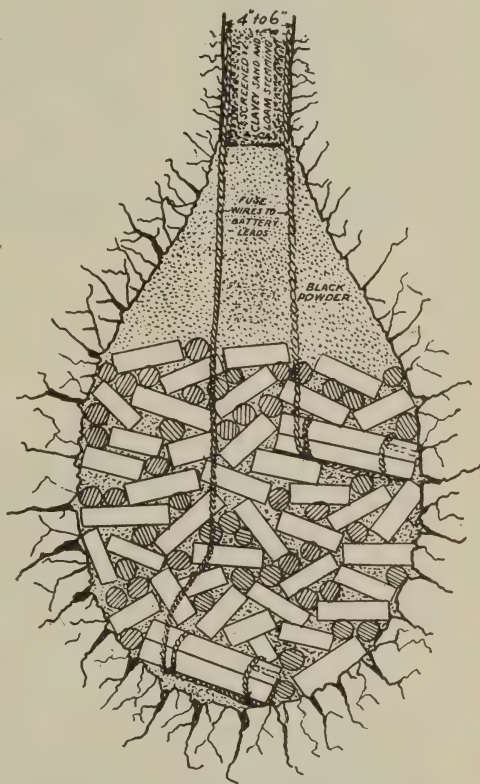


FIGURE 10.—Typical blasting charge in bank hole.

holes, a little at a time, with the aid of a charging stick—a long, slender wooden pole (Pl. VII, A).

In general the equipment of the charging squad is as follows:

"Springing" gage.  
Charging sticks.  
Pocket reflecting glass.  
Galvanometer.  
Rheostat.  
Blasting battery.  
Explosives.  
Fuse.  
Electric detonators.

Combination detonator crimper, pliers, and punch.  
Wooden mallet.  
Large funnel.  
Pocket knife.  
Wooden axe handle.  
Detonators.  
Battery lead wire.  
Connecting wire.

After the holes have been charged the detonator wires are connected, generally in series, with the wires that lead to the battery

(fig. 11). A galvanometer is then attached to the lead wires, and if the whole connection makes a complete circuit the galvanometer needle will be deflected. If there is a break in the circuit it is located by attaching the wires so that first one hole, then two holes, and so on, are left out of the circuit; as soon as a complete circuit is found, it is known that the break must be between the completed circuit and the hole last cut out of circuit. When a complete circuit is established, the wires are ready for connection with the battery. The current

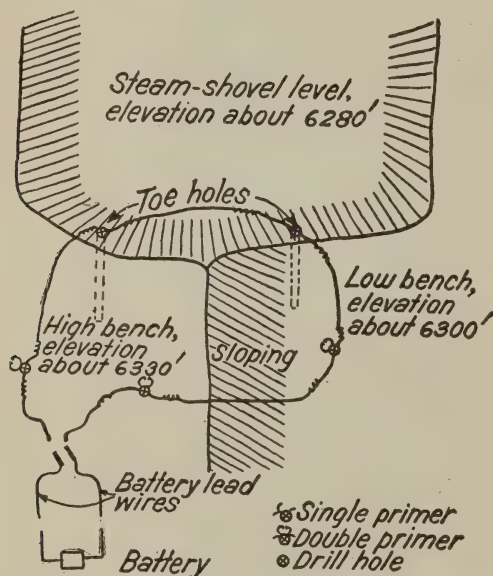
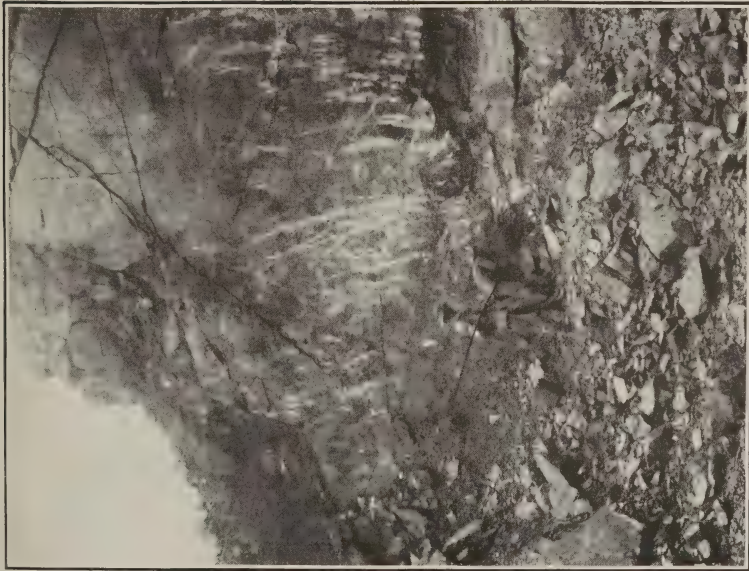


FIGURE 11.—Arrangement of detonator and electric wires for exploding charge.

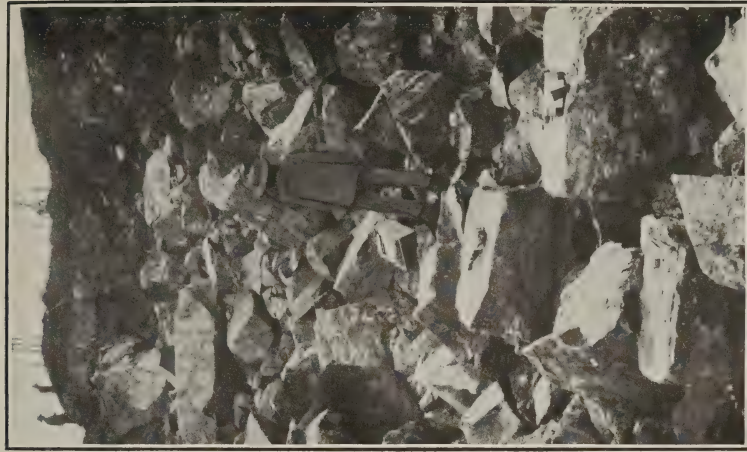
used in the testing galvanometer is about one-thirtieth of that necessary to detonate the charge, so that there is an ample margin of safety.

Five or ten minutes before the charge man finishes connecting the detonator wires he notifies the engineer of the nearest locomotive or steam shovel, who toots the whistle 40 or 50 times. This warning is understood by all workmen, and they immediately retreat to a safe distance. The final connection with the battery is then made, and after an interval of a few minutes the charge man sets off the round





4. LOADING A TOE HOLE. THE CHARGE IS BEING PUSHED IN WITH A LONG SLENDER WOODEN POLE KNOWN AS A "CHARGING STICK."



B. FACE OF SHOVEL CUT AFTER FIRING BLAST HOLES, SHOWING LARGE BOWLDERS ON WHICH "DOBE" CHARGES ARE BEING LAID.



of shots. After the explosion he notifies the steam-shovel or locomotive engineer, who, when he returns to his post, sounds a long blast of the whistle, indicating that the danger is over.

If a charge misses fire, of course no attempt to withdraw it is made. Such a proceeding would be entirely too dangerous. The remedy is to drill a new hole within 6 or 8 feet of the unexploded one. The new hole must not be so close to the old one that deflection may cause it to break into the unexploded charge. The distance is gaged by the charge man who chambered the old hole and knows the size of the chamber, and by the driller who drilled it. During the drilling of the new hole, as a precautionary measure, nobody is allowed to work in the vicinity of the unexploded hole except the men necessary to operate the drill.

#### FACTORS AFFECTING BLASTING EFFICIENCY.

The quantity of material loosened by a given amount of explosive depends on many factors, chief among which are: Toughness of the rock; degree of jointing and fissuring; number and extent of free faces of the bench to be blasted; depth and spacing of the holes—their distance apart and from the free faces; character of chambering and of charging of the holes; character of detonation of the charges—whether they are all detonated simultaneously.

A factor of special significance in determining the charges to be used is the degree of solidity or "tightness" of the rocks at the base or toe of the blasting face. Where the rocks are solid at the toe, toe holes with fairly heavy charges greatly increase the efficiency of the blast, and cause the rock to break down to an even surface. Data regarding several bank and toe holes projected are shown in figure 12. The conditions seemed to be about average for the mine. The results from the blast with these holes were as follows: About 3.2 cubic yards of soil and weathered surface rock was broken per pound

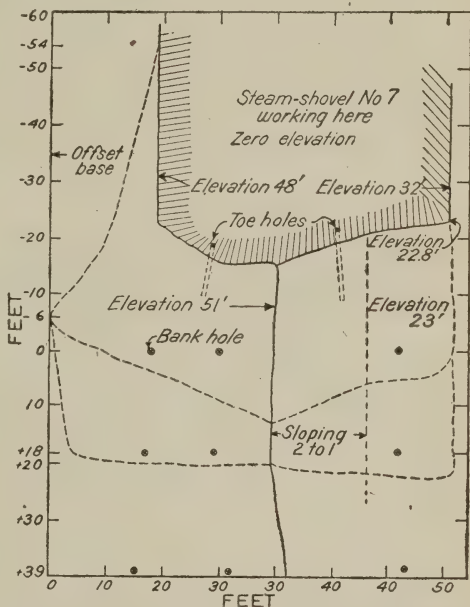


FIGURE 12.—Results of firing bank and toe holes in positions shown. Arrangement of holes and superficial area cut by each series are indicated by dotted lines.



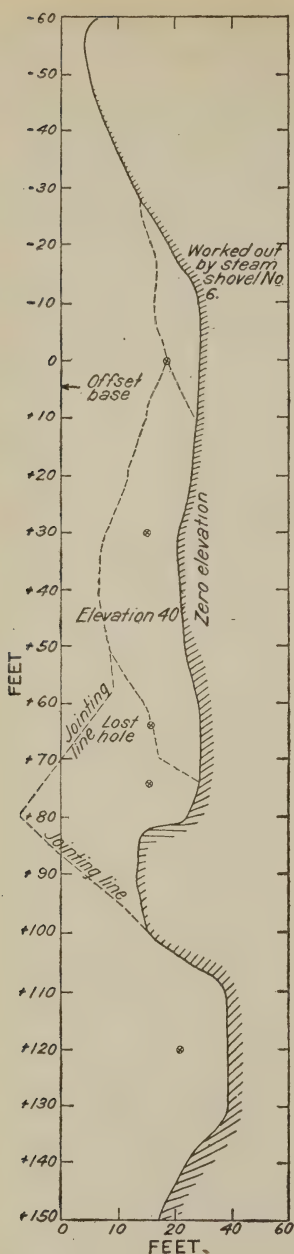


FIGURE 13.—Results of firing blast holes in firm rock. Shows arrangement for single-line bench holes. The areas cut by each are indicated by dotted lines.

of black blasting powder. Approximately 3.1 cubic yards of somewhat weathered dioritic rock was averaged per pound of Trojan powder. Where this rock was firmer and more solid the results averaged about 1.5 cubic yards for a pound of Trojan and 40 per cent strength Repauno gelatin dynamite (fig. 13). In the tougher areas only about 0.4 cubic yard was broken per pound of Trojan powder. Average results from a number of blasts are given in the following table:

*Average results from several blasts.*

Character of rock and ore.	Depth of holes.	Explosives used.	Rock broken.	Rock broken per pound of explosives.
Soil and weathered rock.....	<i>Feet.</i> 50	<i>Pounds.</i> 378	<i>Cubic yards.</i> 1,222	<i>Cubic yards.</i> 3.233
Partly weathered dioritic rock....	70	700	2,168	3.097
Do.....	28 and 50	2,235	3,249	1.454
Relatively soft porphyry.....	40	825	2,264	2.744

For a three-month period in 1913 the average quantity of ore and rock broken per pound of explosives used was 2.55 cubic yards. For a corresponding three-month period in 1914 the average was 2.67 cubic yards. The quantity of both kinds of explosives used during these periods was as follows:

*Quantity of explosives used in blast holes during two three-month periods.*

Kind of explosive.	Three-month period in 1913.		Three-month period in 1914.	
	Quantity of explosive used.	Per cent of total.	Quantity of explosive used.	Per cent of total.
Black powder.....	<i>Pounds.</i> 192,870	44.56	<i>Pounds.</i> 136,950	39.71
Higher explosives.....	240,000	55.44	207,950	60.29

## SECONDARY OR "DOBE" BLASTING.

When the bank holes are exploded, 5 to 15 per cent of the material broken is so coarse (Pl. VII, *B*) that it has to be blasted again in order to be handled easily by the buckets of the steam shovels, which have a capacity of 3 to  $3\frac{1}{2}$  cubic yards. This secondary blasting is done by placing a number of sticks of dynamite and a primer, with 42 inches of fuse attached, on the rock to be broken, the size of the charge being regulated by the judgment of the powder man. Formerly it was the general practice everywhere to cover charges of this kind with a stemming of clay or adobe, giving rise to the term "adobe" or "dobe" blasting. This manner of breaking rocks is also referred to as "bulldozing."

When the "dobe" charges are all set, the nearest steam whistle toots a warning for the workmen to get out of the way. The charge-man and one or two assistants, each using a piece of fuse equal to the length of the primer fuses (42 inches) as an igniting medium, light the primer fuses and then run to safety. A series of cuts half through the igniter fuse indicate to the men doing the igniting just how fast their fuses are burning and how much time they will have to finish the lighting and get to safety. The number of "dobe" shots placed is counted, and the number of explosions heard is carefully observed so that any missed or delayed shots will be detected.

Prior to the installation of the plant for crushing coarse material, the cost of blasting boulders, or of "dobe" shooting, was about 1.54 cents per ton of ore loaded. It is estimated that the new crushing plant saves 80 to 90 per cent of that cost.

## BLASTING COSTS.

During a three-month period in 1913, when the mine was working at normal capacity, the cost of explosives was 3.75 cents per cubic yard of material loaded. For a corresponding period in 1914, when operations were conducted at only about 60 per cent capacity, the charges for explosives had dropped to 3.56 cents per cubic yard. The difference was due largely to the fact that fewer boulders had to be blasted after the coarse crushing plant had been installed.

The average cost per pound of all the explosives used in 1913 was 9.55 cents, and in 1914 it amounted to 9.64 cents. "Dobe" blasting is an inefficient method of breaking rock, because the charge is not confined and much of its energy is lost; also because the loading operations of the nearest shovel or shovels must cease while the "dobe" charges are being placed and fired.

## BLASTING ACCIDENTS.

Under normal mining conditions about 1,500,000 pounds of explosives is used annually by the Chino company. These figures are so large that, in comparison, the accidents have been few. During 1914 only two men were killed as a result of blasting operations. They miscounted the number of "dobe" shots lighted, and walked back before the last one, which was delayed, had exploded. In 1913 three men were killed. A party of five men had taken refuge under the back end of a steam shovel, against the general orders of the company. The blast was so heavy that the concussion moved the shovel back on the men, fatally crushing three of them.

## SAFETY RULES.

To help safeguard its employees the company has posted rules, printed in English and Spanish, so that everyone, especially those that handle explosives, may become familiar with them. Some of the principal rules so posted follow:

Before exploding a blast, care should be taken to give the blasting signal and to wait until every person has reached a point of safety. Do not explode a blast until at least three minutes have elapsed from the last note of warning.

The powder "whistle off" signal must be given after each and every time of blasting.

All persons are warned not to return to the vicinity of "dobes" until at least one-half minute has elapsed from the last shot heard.

When dobbing (bulldozing) is done directly in front of a shovel no one should remain on or under that shovel for protection.

In firing bank shots or toe holes the battery wires must not be connected to the battery until the person immediately in charge walks back to the battery and gives the word to connect.

Do not smoke while handling explosives, and do not handle explosives near an open light.

Do not carry loose detonators or electric detonators in the clothing. Carry them in special boxes.

Do not transport caps or cartridges containing caps to the blasting place with the supply of dynamite to be used, and do not place them side by side until ready for explosion.

Do not tamper with caps. If the caps are damaged, discard by loading into some bank hole.

Inspect the place where bulldozing has been done before work is resumed. Recover all powder and caps possible and return to their proper places.

Carelessness will not be tolerated. The careless laborer imperils not only himself but he endangers his fellow workman as well.

All blasting machines must be tested with rheostat at least twice a week. If blasting machine is defective, this must be reported. The machine must not be used until it has been repaired.

During cold weather no dynamite but the nonfreezing powder shall be used unless by order of the general foreman.

Powder boxes and powder cans must not be opened otherwise than with the wooden tools provided (which are mallet and axe handle).



Black powder cans should not be rolled down the bank.

Black powder should not be opened near a light or fire.

Dynamite must not be rolled down the banks tied to a rope except in places and manner designated by the head powder foreman.

It is absolutely prohibited to store powder or caps anywhere about a steam shovel. All such recovered material must be gathered up and returned to the boxes.

Do not allow explosives to become scattered. Immediately after each finished period of loading all unused powder and caps must be returned to their respective places.

Detonators and powder must not be stored in the same place.

Fuse, firing machines, and all tools necessary for blasting must be kept in a box separate from the powder. All material must be kept under lock and key.

Place all powder boxes so that the cartridges will rest in a level position.

Do not fasten a detonator with the teeth or by flattening with a knife; use a crimper.

Do not explode a charge to chamber a hole and then immediately reload the hole; the hole will be hot. Water must be used to cool it.

Do not put two independent charges of explosives in the same drill hole.

Do not attempt to draw nor to dig out the charge in case of a misfire.

No person shall blast a hole alone; he must have some one near him.

For dobbing, fuse must not be less than 30 inches; for springing holes not less than 18 inches.

All electric detonators must be tested with galvanometer before placing stemming in the hole.

Not more than three men are to load a hole at one time.

In loading bank holes one primer must be placed in the hole before adding any of the charge of powder.

After primer or powder has been placed in the hole no metal shall be used either to tamp or clear the hole.

All tamping used in holes shall be screened dirt.

Caps and electric detonators shall be stored in separate box, which shall be marked "Cap Box." Nothing else shall be stored in these boxes.

One man is never to trim a bank alone; at least two men should work together.

While trimming, bank men are not to go down the bank unless with a rope securely fastened above.

#### "GOPHER" BLASTING.

At the present stage of development "gopher" blasting has been practically discontinued. However, in the past it was advantageously used to break down some high banks that were difficult to drill and load in the ordinary way. A good account of one of the largest gopher blasts has been written by R. I. Kirchman,<sup>a</sup> bench foreman for the Chino company. It is so instructive that extensive quotations from it follow. Kirchman first explains that in the preliminary work at Santa Rita, owing to the uneven surface of the ground, the height of the benches varied from a few feet to 150 feet or more. Some of the high benches were maintained because the expense of opening intervening levels was not warranted by the conditions. Locally the rock was considerably fissured and jointed with a tendency to "hang up." Hanging brows of the rock, where the bank was 75 to 100 feet high, came down at uncertain intervals,

<sup>a</sup> Kirchman, R. I., Gopher blasting: Colorado School of Mines Mag., vol. 4, May, 1914, pp. 103-107.

sometimes endangering the steam shovels near the base. When toe holes were sprung they often caved, thus rendering them useless and necessitating new holes. Because of these difficulties it was decided to handle the bank with a large gopher blast. Kirchman describes the operations as follows:

Three adits were determined upon as shown in figure 14. The north adit was numbered 1, the middle 2, and the south 3. Adits 1 and 2 required two tunnel sets each. These sets were lagged. Adit 3 was not fractured as much as 1 and 2 and did not require timbering. The cross-section dimensions were 3 feet by 5 feet high, driven part by hand drill and part by air-drill machine.

The muck from the main adits was dumped directly in front and around their entrances. The broken earth from the crosscuts was thrown back and leveled to a height just sufficient to allow the passage of a man—that is, an opening about 2 by 3 feet. The excess muck was carried to the entrance. The corresponding crosscuts were thus left half full of muck back to the main adits.

The ends of the crosscuts were chambered to a size sufficient to accommodate the powder predetermined by the amount of yardage over the "gophers." The gopher was assumed to break from a line 10 feet beyond the breast of the main adits, and along a 1-to-1 slope to the surface.

The Mexican laborers were divided into classes of miners and muckers. The miners were paid at the rate of \$2.75 per day of 9 hours and the muckers \$2 per day. At the start only two 9-hour shifts were employed, but as greater progress was desired three 8-hour shifts were put on.

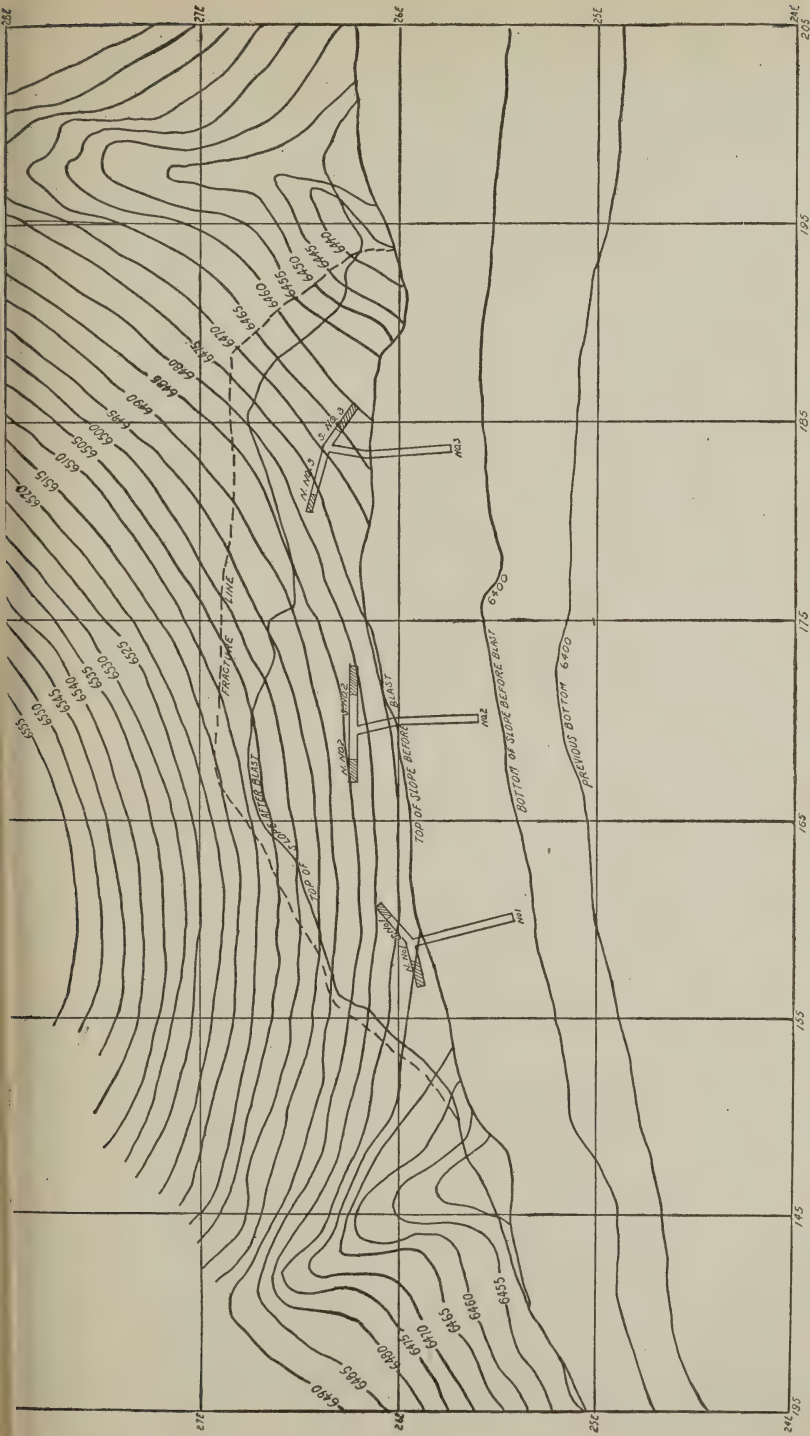
The latter change left 2 men in each face, or a total of 18 men per 24 hours. The progress was about 4 feet per shift. The total linear feet of gopher driven was 352. The time occupied in driving was 31 days. Loading commenced immediately after completion and required 3 days of 10-hour day shifts, and was accomplished by 4 powdermen and 10 laborers for loading and tamping.

On the bottom of the drifts a walk was laid, composed of single boards placed loosely end to end; no nails were used. The powder boxes were slid along this path to the chambers. Ordinary pocket electric flash lights gave fair light for about 24 hours.

Crosscut 1 was loaded first. The ends of the powder boxes were opened and placed end to end. The boxes were pyramided in a single tier down the middle of the pocket. An electric fuse was placed in the middle box of each layer of the tier, making three fuses to the chamber. Extreme care was taken not to injure the light fuse wires. An important precaution to observe in loading these initial charges is the avoidance of undue pressure upon the fuses or stick powder. The stick powder used was Du Pont 40 per cent Repauno gelatin. The fuses were Victor No. 7, 10-foot lengths, having a resistance of 1.2 ohms.

The black powder, in 25-pound cans, was next placed about the powder boxes. About three-quarters of the black-powder charge was loaded from the breast to the front of the charge and one-half way up to the back. Judson powder, in 12½-pound paper bags, was placed upon the powder cans as compactly as possible. This latter charge filled the pocket clear to the back. The final face of the chamber was sealed with the remaining one-quarter of black powder in cans. The latter precaution was taken to avoid any danger or damage to the dynamite and Judson powder when tamping.

All connecting fuse wires were firmly contacted. The charges were connected in series with No. 14 lead wire, and the fuse wires were wrapped around the lead wires. All fuses were tested with a Du Pont galvanometer, which shows the presence of a circuit and resistance contained therein. All completed circuits in the different chambered pockets were tested before the chambers were sealed with carefully placed





muck. The lead wires from each chamber were laid on top of the muck and in one corner of transverse cross section. Pieces of wood broken from the refuse powder boxes were placed at an angle above these wires, thus affording a protection from the rocks. These lead wires were uncoiled as fast as the tamping proceeded.

The tamping was a slow and laborious process. The muck had to be carried in powder boxes and slid along the board path, and dumped into the receding filled faces. At intervals of 3 feet of fill the circuits were tested in all drifts.

During the loading of the gopher the men suffered from powder headaches. These were sharp and annoying, but soon passed away in most cases when brought to the fresh air. The tamping progressed more rapidly when the main adits were reached. An unusual source of danger occurred for a few days during the presence of electrical storms. The precaution of keeping all lead wires well within the adit entrances was observed.

The lead wires from the several entrances were then connected in series. A pair of main lead wires was then laid a distance of 1,500 feet north of the entrance of adit 1. These were not connected until the proper time for blasting. The blasting battery, a Du Pont 1-75 fuse machine, was cleaned and tested.

The final circuit was tested with the galvanometer and a resistance of 24 ohms noted; this indicated a good connection. The eighteen 10-foot fuses used should have had a total of 21.2 ohms; the remaining 2.8 ohms was apparently due to imperfect connections. The resistance of the lead wires was practically zero. On June 25, 1913, the gopher was fired. Some boulders were thrown 400 to 600 feet by the concussion of the sharp blast. The main mass of the material was not thrown over 200 feet.

The yardage of the gopher was figured on a 1-to-1 slope. The steam shovels removed 60,000 cubic yards (in solid), but the total has not yet been removed. It is estimated that 40,000 cubic yards, in solid measurement, of loosened material is still to be removed.

It was the opinion of some that the gopher was loaded a trifle too heavy. For the class of overburden a load of 1 pound of powder for 3 or 3½ cubic yards (solid) would have yielded better results; but as an experiment for future reference this was a successful gopher. No serious delay occurred, no damage was done, and the practicability of a gopher bank was demonstrated.

The data were summarized by Mr. Kirchman as follows:

*Data regarding one of largest "gopher" blasts used by Chino company to break down high bank.*

Chamber.	Powder loaded (pounds).					Rate loaded.		
	Black powder.	Judson.	Forty per cent gelatin.	Ratio of explosion to ignition charge. <sup>a</sup>	Total powder charge.	Engineers' estimate.	Pounds per cubic yard.	Cubic yards broken per pound of explosives.
N. No. 1.....	7,725	2,400	800	12.65	10,925	<i>Cu. yds.</i> 25,664	0.41	2.34
S. No. 1.....		5,500	500	11.00	6,000	14,948	.40	2.49
N. No. 2.....		5,250	600	8.75	5,850	18,122	.32	3.10
S. No. 2.....		9,250	800	11.56	10,050	25,197	.40	2.50
N. No. 3.....		7,050	500	14.10	7,550	12,446	.67	1.65
S. No. 3.....		9,050	900	10.05	9,950	22,283	.45	2.24
Total or average..	7,725	38,500	4,100	11.27	50,325	118,660	.425	2.36

<sup>a</sup> These data are of interest in that they indicate the proper ratio of the ignition charge to the explosion charge. An idea of the ratio of the overburden in cubic yards to the powder in pounds is suggested. A large ratio in this character of ground is recommended.

## LOADING ORE.

After the ore and waste have been broken by blasting they are loaded onto cars by Marion steam shovels. Of these, there are six of the 91-ton model, two of the 92-ton model, one of the 100-ton model, and one of the 40-ton model—ten in all.

The 40-ton and the 100-ton shovels are used for digging first or pioneer cuts, also called "thorough" cuts. The 40-ton machine has a large arc of swing to its boom, so that it can carry a cut into a bank, swinging its dipper back to one side far enough to dump into a car close to the side of the shovel and back from the working face of the cut. The other shovels when making first, or pioneer cuts, have to load onto cars that stand on the bank above, outside the excavation that is being made. The 91-ton and the 92-ton shovels are designed to have a maximum lift of 19 and 21½ feet, respectively; hence, in general, they can not make a first cut more than 10 to 12 feet below the level of the loading track. The 100-ton shovel, however, is built with a long boom and dipper arm, so that it has a maximum lift of 35 feet. Its function is primarily to make pioneer or "thorough" cuts, which the other shovels enlarge. It can cut a wide ditch about 22 feet deep, lifting the material up onto cars that stand on the banks outside the cut. It has a dipper with a capacity of 3 cubic yards, whereas the capacity of the 91-ton and of the 92-ton shovels is 3½ cubic yards. This smaller dipper is necessary because the longer boom and dipper arm—92-foot extreme radius—greatly increases the strain on the machinery.

The crew of each shovel consists of an engineer at \$190 per month, a craneman at \$135 per month, a fireman at \$3.50 per 10-hour shift, an oiler at \$2.25 per shift, and 6 to 8 pitmen at \$2.25 per 10-hour shift, who assist in loading operations and in moving the shovel.

The material handled per day of two 10-hour shifts by each shovel, except the small 40-ton machine, is 1,400 to 2,800 cubic yards, measured in the solid. This rather wide variation is due to occasional shortage of cars, local difficulties of digging, or temporary breakdowns.

The coal, oil, and cotton waste used per 10-hour shift is as follows:

*Coal, oil, and cotton waste used per 10-hour shift.*

Material.	Price.	Amount used by different sized shovels.		
		40-ton.	91-ton and 92-ton.	100-ton.
Lack oil.....	\$0.14 per gallon....	3 pints.....	2 quarts.....	2 quarts.
Cylinder oil.....	.37 per gallon....	3 pints.....	3 quarts.....	1½ gallons.
Engine oil.....	.22 per gallon....	½ pint.....	1 pint.....	1 pint.
Rude oil.....	.10 per gallon....	1 quart.....	2 quarts.....	1½ gallons.
Sp grease.....	.05 per pound.....	½ pound.....	½ pound.....	½ pound.
White cotton waste.....	9.39 per hundred-weight.	1 pound.....	1 pound.....	1 pound.
Run-of-mine coal.....	4.30 per ton.....	3½ tons.....	5½ tons.....	7½ tons.

Heavy castings and repair parts for all the shovels are kept on hand so that a shovel can, without much delay, be almost rebuilt in the company's machine shops.

The cost of loading is in general 6 to 12 cents per ton. In a few cases, owing to exceptionally unfavorable circumstances, the loading costs for one shovel for a month have averaged as high as 29 cents per ton. On the other hand, exceptionally favorable conditions have at times reduced the monthly average to a little more than 4 cents per ton.

#### TRANSPORTATION OF ORE.

In the matter of transportation, it was planned to have the loaded cars moved from and the empties moved to the shovels with as little delay as possible. To handle the trains of ore and waste fourteen 42½-ton and seven 50-ton, 4-wheeled Porter locomotives are used. These have cylinders 15 by 24 and 16 by 24 inches. They are each manned by a locomotive engineer, at \$4.25 per 10-hour shift and a fireman, at \$3 per 10-hour shift. The quantities of oil and fuel used by each locomotive per shift of 10 hours are about as follows:

*Quantities of oil and fuel used by 42½-ton and 50-ton Porter locomotives with side or saddle water tanks.*

Engine oil.....	1½ pints, at 22 cents per gallon.
Cylinder oil.....	1 pint, at 37 cents per gallon.
Cup grease.....	½ pound, at 5 cents per pound.
White cotton waste.....	½ pound, at \$9.39 per hundred-weight.
Run-of-mine coal.....	3 tons, at \$4.30 per ton.

In general, there are always about two locomotives in the shop undergoing repairs. In addition, one locomotive is held just outside the shop with steam up ready for instant service. The total repair, upkeep, and all other expenses of the locomotives amount to about 0.2 cent per ton of material handled.

The ore cars in use are about as follows: Forty-five 12-yard, side-dump, Oliver steel cars; forty 6-yard steel cars not equipped with air brakes; four 20-yard steel cars fitted with air brakes, air dumping appliances, etc. The four last-mentioned cars are leased from the Santa Fe Railroad, and have given such good satisfaction that the Chino company has ordered several of the new Clark 20-yard extension dumping steel cars (Pl. VIII). These dump on either side, by air or by hand power, and are said to be satisfactory.

The experience of the Chino company has been that 6-yard cars are too light for loading with heavy shovels and too expensive to keep in repair. Not being fitted with air brakes, they are rather dangerous to handle on heavy grades. The large, heavy, air-brake





A. STEEL ORE CAR USED AT SANTA RITA. DUMPS ON EITHER SIDE BY AUTOMATIC AIR CONTROL OR BY HAND, CAPACITY 20 CUBIC YARDS.



B. STEEL ORE CAR IN DUMPING POSITION.



and air-dump cars, though expensive, are cheaper in the long run, under the conditions of heavy service that prevail at Santa Rita.

Ore that is fine enough not to need crushing is generally loaded directly into 50-ton steel railroad cars, the property of the Santa Fe Railroad Co., and is sent direct to the mill at Hurley without having to pass through the crushing plant just below the mine.

The following table shows the number of cars of each size loaded per month, during five typical months, and the cost of keeping them in repair. The repair cost per dry ton of ore handled and per cubic yard of waste is also given. The 50-ton cars are the property of the Santa Fe Railroad Co., but are kept in repair by the Chino company while used by them, and so are included in the repair account.

*Cost of car repairs for five typical months.*

CARS USED IN MINING ORE.

Month.	Cost of repairs.			Number of cars loaded.			
	Labor.	Supplies.	Cost per dry ton.	20-yard.	12-yard.	6-yard.	50-ton.
1914.							
January.....	\$188.81	\$196.19	\$0.00196	-----	100	-----	4,028
February.....	148.99	85.01	.00124	-----	4	-----	4,066
March.....	138.48	103.40	.00109	-----	898	-----	4,435
August.....	94.93	78.72	.00142	107	2,103	-----	1,837
September.....	86.24	91.87	.00171	12	2,015	-----	1,438

CARS USED IN STRIPPING WASTE.

			Cost per cubic yard.				
January.....	\$1,699.33	\$1,765.73	\$0.01122	-----	27,124	15,711	-----
February.....	1,340.88	765.08	.00741	-----	29,573	13,043	-----
March.....	1,246.36	930.61	.00629	-----	34,525	16,105	-----
August.....	854.35	708.48	.00838	1,017	10,169	10,778	12
September.....	776.17	826.86	.01008	1,548	9,511	9,065	14

The material loaded onto cars by the steam shovels is hauled out of the pits to one of the following destinations: The mill, direct; the crusher; the ore stock pile; or directly to the smelter. The waste goes to the waste dumps. The lowest grade of ore, containing about 0.8 per cent copper, which has to be moved to get at other higher grade ore and to develop the workings properly, is sent to the ore stock pile if the copper market happens to be unfavorable. It is left there until such time as it can be most favorably utilized. Only the ore that contains many boulders too large to pass the 12-inch by 13-inch grizzly at the mill is sent to the crusher. The other ore is sent direct to the mill, except the occasional carloads that average high in copper; these are shipped direct to the smelter.

The grade of the pit tracks varies considerably. The average grade of all the tracks to the waste dump, the stock pile, the crusher,



and the railroad yards is estimated to be  $2\frac{1}{4}$  per cent. Some idea of the amount of material handled over the company's tracks during the month of July, 1914, when the mine was working at normal capacity and during the month of September, 1914, when it had greatly curtailed its output owing to the European war, is given in the table following:

*Data showing material handled on Chino company's tracks.*

Month.	Ton-miles of haul.					Grade miles, haul to waste and ore piles and to crusher.
	Waste.	Ore piles.	Crusher.	Mill or smelter.	Total.	
July..... 1914.	953,055	39,911	.....	228,771	1,221,737	2,533
September.....	447,486	3,045	36,517	78,830	565,878	2,055

The figures in the second, third, fourth, and fifth columns of the above table are obtained by multiplying the average length of haul by the tonnage moved, thus giving the ton-miles; the figures in the last column are the product of the linear miles and the rise in feet per mile for hauls to the waste and ore piles and to the crusher. The cost of hauling out and dumping waste averaged a little less than 3 cents per cubic yard.

The tracks are of standard gage and consist of 60-pound rails laid on 7-inch by 9-inch ties 8 feet 2 inches long and spaced 20 ties for each length of 33-foot rails. A few 75-pound rails are used in the pits. The Chino company constructed at its own expense and is operating about 24 miles of railroad track. This trackage connects with the railroad yards of the Santa Fe branch line to Santa Rita. It is provided with ample turnouts and crossovers and is systematically operated under the supervision of a yardmaster and his assistants, switch tenders, etc.

The maintenance of the pit tracks is under the supervision of an assistant roadmaster. Subordinate to him are 4 track foremen, each in charge of a gang of 13 men. These gangs shift track and keep the track lines up to the steam shovels. The waste and ore-dump tracks are looked after by 7 track foremen, each in charge of a gang of 13 men. These crews level off and trim down the dumps and keep the tracks shifted out toward the edge of the dump slope. The pay of the laborers is about \$2.05 per 10-hour shift. The track foremen receive \$4.55 per 10-hour shift.

#### CRUSHING OF COARSE ORE.

The coarse ore grizzlies at the Hurley mill have openings 12 by 13 inches. It is intended that the ore shipped to the mill shall have few fragments too coarse to pass through these openings. Up to 1914

ore fragments coarser than this had to be broken by "dobe" blasting, or by hand, either at the shovel, on the cars, or at the mill. These methods of breaking the large fragments were relatively expensive. It was therefore decided to build near the mine a plant for crushing coarse ore, the plant to accommodate any boulder that could be handled by the 3 to 3½ cubic yard dippers of the steam shovels.

A site for the new crushing plant was selected on a side hill, so that the ore cars could be run on a track about 50 feet higher than the feeding hopper of the large crusher (Pl. IX, A). It is situated about 1½ miles below the mine, and is convenient to the branch line of the Santa Fe Railroad which leads to the mill at Hurley. The cars of coarse ore from the mine are dumped onto the steel grizzlies (Pl. IX, B), the bars of which are about 8 inches apart. The oversize pieces roll down into a hopper that feeds to a 48-inch by 60-inch Superior jaw crusher. This vast crusher weighs 100 tons, and is driven by a 150-horsepower motor. It crushes to 6 inches and smaller, and has a capacity of 3,000 to 5,000 tons per 10 hours. A 48-inch overlapping steel pan conveyor, driven by a 50-horsepower Westinghouse induction motor, conveys the crushed product, and the fines that have gone through the grizzly bars (Pl. IX, B), up a 15½° incline to the top of the loading bin (Pl. IX, A). A number of observations showed that one-fortieth to one-fourth of the contents of each car dumped onto the grizzly went over into the crusher. The remainder went through the bars as undersizes or fines.

The loading bin, of steel, is 40 by 14½ feet at the base, and 23 feet high. The upper 9 feet of the longer dimension tapers in to 26 feet 8 inches instead of 40 feet as at the base. It has a capacity of 567 tons, on the basis of 21 cubic feet to the ton of ore. The bin has six loading chutes on each side, and is between two tracks, so that two large steel cars can be loaded at the same time. The loading chutes are high enough above the rails so that the loading is all by gravity. From 10 to 20 minutes is required to load a car having a capacity of 100,000 pounds. The coarser the material the easier the loading, after it is once started. Fine material sometimes chokes the chutes, until prodded loose by the bars of the car loaders.

About four 12-yard cars at a time are pushed by a dinkey locomotive from the mine to the up-grade side of the grizzly. The locomotive engineer then gives three blasts of his whistle. If the crusher man is ready for more ore he answers with two whistles. The first car is set opposite the grizzly and dumped by hand, then the other cars are set and dumped as fast as the crusher can take the ore. After all the cars in a trip have been dumped, the locomotive engineer sounds two blasts of his whistle, and pulls the dumped cars past the grizzly; after the cars have been uprighted and fastened, he proceeds to the mine for more material.

Five men are required at the top of the grizzly to dump and upright the cars. A trip of 4 cars can be dumped and uprighted in about 5 minutes. After the train has pulled out, the men shovel onto the grizzly any material that has fallen aside. The crew operating the crusher consists of a foreman, two mechanics, two crusher runners, four car men, one loader, and three laborers. The total operating wages amount to about \$35 per 10-hour shift.

The total cost of building and equipping this crushing plant was about \$127,000, of which \$42,000 was for labor and \$85,000 for supplies.

The total cost of operating the plant for crushing coarse material for a part of August, 1914, was \$1,231.84, or at the rate of  $1\frac{1}{4}$  cents per dry ton of ore. For September, 1914, the total operating cost of the plant was \$1,994.84, or at the rate of 1.9 cents per dry ton of ore. This higher cost was due to some important reinforcements required for the retaining walls and additional equipment charges.

#### MINE DRAINAGE.

The precipitation in the Santa Rita region is so light that the amount of water that gets into the workings is relatively small. The slopes in some of the open cuts are such that they drain directly to the Santa Rita Creek. In the deeper open cuts sumps with electrically driven pumps take care of the water. The 300-foot level of the underground workings, which are old workings kept open mainly for drainage and prospecting, is drained by two Aldrich triplex pumps and one Byron-Jackson centrifugal pump, all operated by electric power. These pumps have a total capacity of about 1,000 gallons per minute, against a head of 400 feet. A reserve pumping plant on this level consists of two Cameron pumps, 16 by 7 by 18 inches. They are operated by compressed air, and are held in readiness in case of accident to the electric pumps or to the supply of electric power. On the 400-foot level the small amount of seepage is collected in a sump, and pumped to the 300-foot level by a 16-inch, three-stage, electrically driven Byron-Jackson centrifugal pump, having a capacity of 900 gallons per minute.

#### EFFECT OF SLOPES OF OPEN CUT.

The stability of the slopes of an excavation has an important bearing on the cost of the excavation. Where the slopes stand steeply the cost is much less, other things being equal, than where they slide to flat angles, owing to softening by ground water or other causes. If the slopes of the excavations at Santa Rita were as unstable as, say, those of the Culebra Cut, much would be added to the cost of mining.





A. PLANT FOR CRUSHING COARSE ORE, AND LOADING BIN, SHOWING CARS AND LOADING TRACKS.

The coarse material is hauled to this plant over the upper track, the grade of which is visible in the upper part of the picture, and dumped over the grizzly, from which the oversize goes to the crusher situated in the structure below and in front of the grizzly. An inclined belt conveyor with overlapping steel pans elevates the fines from the crusher and grizzly to the loading bin, from which the cars are loaded on the lower tracks.



B. VIEW OF PART OF THE GRIZZLY SHOWN IN PLATE IX, A. THE OPENINGS BETWEEN THE STEEL BARS ARE ABOUT 8 INCHES WIDE.



At Santa Rita great open cuts have been made and from these ore and waste are being excavated by steam shovels. These vast open pits are thus constantly being widened, from the central area outward, and are being deepened to the bottom of the ore body. The steam shovels operate on benches, as already explained, one or more to a bench, depending on the location of the ore bodies, the requirements of mining, etc. The bench levels, in this process of being widened, are cut back into the rising ground, and into the sides of the hills, necessitating slopes that, locally where ridges are cut, are several hundred feet high.

These high slopes, in places, may be largely in waste material that overlies valuable ore bodies; hence their stability has much to do with the cost of mining. If they stand steeply, much less excavation is necessary than if they slough down to flat grades. At Santa Rita the open-cut slopes have stood remarkably well, chiefly because of the following factors:

(1) There is little ground water in the soil, which covers the rocks to depths of a few inches to 30 feet or more. This is because of the relatively small precipitation and the good drainage conditions. The latter are due to the porosity of the soil and the slope of the ground toward the open cut.

(2) The upper or weathered zone, which varies much in thickness, depending on the character of the rocks, the degree of jointing, etc., is considerably cut by jointing (Pl. X), and some shearing. This weathered zone, together with most of the mineralized rocks, is locally much altered, not only by weathering, but also by hydrothermal action. The effect on the slopes is as follows:

(a) The weathering and mineralizing solutions have much sericitized the feldspars, thus weakening the rocks and causing a considerable degree of pulverization when they are blasted. With excessive ground water this condition would also tend to give them muddiness and mobility. However, ground water being largely absent, as already explained, they are strong enough to stand at fairly steep slopes without crushing or movement, except some sloughing, for all heights of slope that are likely to be encountered.

(b) The jointing has cut the rock mass into irregular blocks (Pl. X), and these will, of course, slough off where the slopes are rather steep, especially after they have been somewhat loosened and jarred by blasting. However, this jointing is not great enough to give a flowage-like motion, or "creep," to the jointed material, even on the highest and steepest slopes. Where faulting has sheared the rocks, and slickensided them, there is considerable local sloughing, but such zones are too restricted to cause a noticeable increase of mining cost.

The rocks below the weathered zone are less jointed and are more stable than those just described for the weathered zone; hence they stand fairly steeply.



The steepest slopes of the cut are those in the quartzite and other hard rocks. They range from  $55^{\circ}$  to  $85^{\circ}$ , depending on whether the principal joint planes dip toward the excavation. Probably the average slope in such rocks is about  $70^{\circ}$ , with the soil covering and loose rock on top sloping at about  $40^{\circ}$ . In the softer rocks slopes of  $40^{\circ}$  to  $70^{\circ}$  are maintained. In calculating the yardage of stripping to be removed, a slope of 1 on  $\frac{1}{2}$  was first used, meaning one unit vertical to one-half unit horizontal. However, in 1915 the use of a slope of 1 on  $\frac{3}{4}$  was adopted as being safer for making ore estimates.

The flattest slopes are those at which the material in the waste dumps comes to rest. Several of the large dump slopes, some showing about 200 feet of elevation were measured, and they all approximated  $37^{\circ}$ , in spite of the fact that they contained much fine material. The upper half of each of the dump slopes was about  $37\frac{1}{2}^{\circ}$  and the lower half about  $36\frac{1}{2}^{\circ}$ . The increased flatness below is due to the tendency of the larger fragments to roll down and flatten the slope at the bottom.

#### COMPRESSOR PLANT.

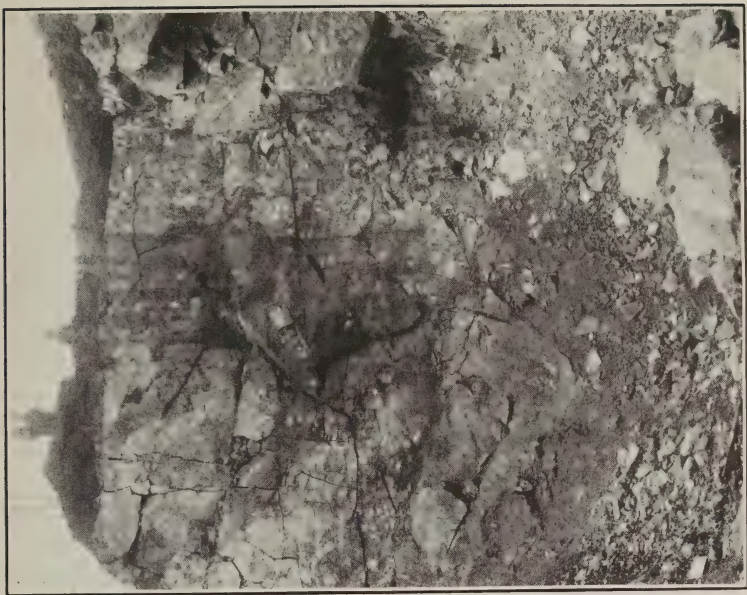
The air-compressing plant contains two Nordberg compound compressors, each driven by a 200-horsepower electric motor. The air, when compressed to 90 pounds per square inch, is stored in two steel cylindrical receivers  $4\frac{1}{2}$  feet in diameter and about 13 feet long. One of these is a discarded steam boiler that serves its purpose reasonably well; the other was built as an air storage tank. From these reservoirs pipes distribute the air to the places in the workings where it is needed to operate the tripod and small hammer drills, and the few churn drills that formerly used steam but now use air. The power charges against the compressor are \$550 to \$650 per month.

#### CENTRAL POWER PLANT AND POWER DISTRIBUTION.

The Chino company has its power-generating and power-distributing center at Hurley, where the concentrating plant is situated. Power is distributed in the form of electrical energy generated in a large modernly equipped steam-driven plant. When the plant was visited by the authors in 1914, steam was supplied from eight Heine safety boilers, each of 445 horsepower capacity. One or two of these were always in reserve, or being cleaned, or undergoing repairs, while the others were in active operation. Two additional 419-horsepower boilers were being added. The boilers were equipped with superheaters, and with mechanical stokers of the Greene and American types. They used natural draft, except the two new ones, which had forced draft.



*A.* FACE OF EXCAVATION, SHOWING JOINTING.



*B.* ANOTHER VIEW SHOWING JOINTING.





The plant operated with steam at a pressure of 180 pounds and heated to 370° F. One fireman and two helpers, per eight-hour shift, operated the plant. The additional boilers called for two more helpers per 24-hour day. In the boiler room were also two boiler washers and one repairman. Feed water heated to a temperature of 190° to 195° F. by Stilwell heaters, and measured by V-notch weirs, is supplied to the boilers by three automatically controlled pumps, each 8 by 12 by 7 by 12 inches.

The boilers supplied steam to three Nordberg compound engines, each 27 by 60 by 48 inches. Each engine drove a 1,250-kw., 480-volt, Allis-Chalmers generator. Steam was also supplied to a 150-horsepower Ballwood tandem compound engine, which drove a 100-kw., 250-volt, Allis-Chalmers generator. Another 100-kw., 250-volt, generator, of the Westinghouse type, was driven by a 150-horsepower Westinghouse steam turbine, and one 100-kw., 250-volt, Allis-Chalmers generator was driven by a 150-horsepower Allis-Chalmers alternating-current motor. There were other smaller motors about the plant. A Worthington surface condenser, operated by steam-driven dry-vacuum and motor-driven wet-vacuum pumps, provided water for each dynamo engine.

The water from these condensers, the condensate, was pumped into a series of baths, in which the cylinder oil and graphite was removed. The cleaned water was then returned to the feed-water storage tanks in the boiler house. The exhaust from the other engines went direct to the feed-water heater. The water for cooling the condensers was circulated by two 16-inch Worthington volute pumps driven by 200-horsepower Allis-Chalmers induction motors operated at 690 revolutions per minute against a pressure of 37 pounds. Only one of these pumps was used at a time, the other being held in reserve.

During July, 1914, the cost for the coal used at the Hurley plant was 33.2 cents per 1,000 pounds of steam generated, and during September it was 32.7 cents.

In the transformer room the plant contained two banks, each of three single-phase, 400-kilovolt-ampere, 60-cycle, Allis-Chalmers transformers, oil-insulated and water-cooled. These transformers could be run in multiple or separately, and they stepped the current from 480 to 24,000 volts.

Attached to the power plant there was an oil-filtering plant for treating used oil. It had a capacity of 25 gallons per minute and a storage capacity of 450 gallons. The used oil drained by gravity to the filtering room and three small double pumps circulated it to the pressure filter tanks. After filtration, the oil was forced by air pressure to the engine room for reuse. All oil pipe lines leading to

and from the filter room were colored. For example, yellow indicated engine oil, brown indicated the cylinder-oil drain to the filter, green the crank-pin oil drain, etc. The cylinder oil required two filterings. The cotton waste was cleaned by a centrifugal machine.

During the year 1913 the power-generating plant had a net output of 24,198,633 kilowatt hours. The average cost for the year for current delivered to the feeder circuits on the switchboard was \$0.012 per kilowatt-hour, not including the distribution losses. An analysis of this cost is given in the following table:

*Analysis of cost of generating power at Santa Rita plant in 1913.*

Item.	Total cost.	Cost per kilowatt-hour.
Overhead expense, direct supervision.....	\$6,081.68	\$0.000251
Labor, operating.....	27,970.48	.001156
Labor, repairs.....	8,864.88	.000366
Supplies, operating.....	167,649.56	.006928
Repair materials.....	4,597.14	.000190
Total direct cost.....	215,163.74	.008890
Overhead expense, general administrative (estimated).....	7,600.00	.000314
Taxes (estimated).....	5,530.00	.000229
Fire insurance (estimated).....	1,383.00	.000057
Interest on plant, \$553,000, at 6 per cent.....	33,180.00	.001371
Depreciation, 4 per cent (amortization fund based on 15-year life, 6.66 per cent depreciation at 6 per cent interest, compounded semiannually).....	22,120.00	.000914
Gross cost of generation.....	284,976.74	.011776

During the month of September, 1914, the Santa Rita operations consumed 105,529 kilowatt-hours. The distribution of power for the month of July, 1914, when the mine was operating at normal capacity, and for September, 1914, when it was operating at about 50 to 60 per cent capacity, was as follows:

*Distribution of power in Santa Rita operations during two months.*

JULY, 1914.

Item.	Kilowatt-hours.	Electrical horsepower-hours.	Kilowatts.	Electrical horsepower.
Milling.....	1,521,101	2,039,011	2,044.49	2,740.60
Pumping.....	478,514	641,439	643.16	862.15
Current for town of Santa Rita.....	202,246	271,107	271.84	364.39
Shops.....	12,150	16,286	16.33	21.89
Lights.....	30,560	40,965	41.07	55.06
Total.....	2,244,571	3,008,808	3,016.89	4,044.09

SEPTEMBER, 1914.

Milling.....	981,331	1,315,459	1,362.97	1,827.04
Pumping.....	342,971	459,746	476.34	638.54
Current for town of Santa Rita.....	105,529	141,459	146.57	196.47
Shops.....	12,150	16,286	16.87	22.61
Lights.....	22,913	30,714	31.82	42.65
Total.....	1,464,894	1,963,664	2,034.57	2,727.31

## SANTA RITA POWER AND SHOP EQUIPMENT.

A high-tension transmission line from the company's electric power station at Hurley, about 11 miles south, furnishes power to operate the shop machines, compressor, plant for crushing coarse material, pumps, etc., and to light the town of Santa Rita. The line is designed to carry a pressure of 24,000 volts, but this high pressure is stepped down by transformers at Santa Rita to 440 volts for use in motors and to 110 volts for lighting.

In general, the repair plant may be said to consist of a machine shop, with blacksmith and boiler shops attached, and a carpenter shop. At the time of the authors' visit in 1914, the working force of the mine had been considerably reduced, owing to war conditions. At that time the machine shop employed 3 blacksmiths, 4 blacksmith helpers, 1 drill sharpener and helper, 9 machinists, 5 machinist's helpers, 4 boilermakers, 4 boilermaker's helpers, and 9 car repairers. The blacksmiths and machinists were paid \$4.50 for 9 hours and the boilermakers \$5. The car repairers got \$3 to \$4 for 9 hours. All the helpers were paid \$2.50 to \$3 for 9 hours.

The shop equipment was as follows:

*Shop equipment of Chino Copper Co. in 1914.*

1 Sellers power hammer, operated by air.	3 emery wheels: 1 wet, 1 dry, and 1 drill grinder.
1 coke furnace for extra-heavy forging.	1 American planer.
2 blacksmith forges.	1 Long & Allstatler punch and shears.
3 lathes: 1 Davis, 12-inch; 1 Bradford, 18-inch; and 1 Pond, 36-inch.	1 Watson Stillman portable wheel press.
1 Acme bolt cutter.	1 6-ton hand crane.
1 Williams pipe machine.	1 power hack saw.
1 Barnes drill press.	2 Buffalo forge blowers, operated by a
1 Miles radial drill.	Allis-Chalmers 25-horsepower motor.
1 Steptoe shaper.	

In the drill-sharpening shop, a No. 3 Leyner drill-sharpening machine, operated by two men, took care of all the steel for the tripod drills. In this shop there was also a boiler flue scaling machine, an oil furnace for heating flues, a flue welder, a flue swedging machine, and a circular saw for cutting wood, all driven by a 15-horsepower motor.

Before the water-softening plant was installed the locomotive and steam-shovel flues had to be taken out and the scale removed about every three months, at great expense. Subsequently the frequency of scaling has been reduced to about once each year. Flues are welded, swedged wherever necessary, and used over again. The composition of the boiler scale taken out was about as follows:



*Composition of boiler scale from flues of Chino Copper Co.*

Constituent.	Per cent.
Volatile matter.....	33
Insoluble.....	13
Al <sub>2</sub> O <sub>3</sub> .....	6
Fe <sub>2</sub> O <sub>3</sub> .....	14
MgO.....	1
CaO.....	28
CO <sub>2</sub> .....	3
S.....	2
Total.....	100

Locomotive, steam-shovel, drill, and pump parts were kept in stock. Car axles, car wheels, and all castings were received in the rough and were machined in the company's shops when needed. Locomotives and steam shovels were practically rebuilt on the premises when necessary.

The carpenter shop, a large building, had a lumber yard attached in which about 100,000 feet of material was always kept on hand. A 25-horsepower motor drove a circular saw, a handsaw, a swing cut-off saw, a lathe, and other accessories. Nine carpenters and four helpers were employed in 1914. Carpenters were paid \$4.50 per day of 9 hours and their helpers up to \$3 per day. Mexican laborers working around the carpenters' shop got \$2 per 10-hour shift. The construction and repair of all the company's dwelling houses and other buildings, as well as bridge building, timber framing, etc., came under the charge of the foreman carpenter.

## POWER COSTS.

During the month of September, 1914, after the mine work had considerably slowed down, 105,529 kilowatt hours of electrical energy was used at Santa Rita at a total cost of \$1,075.31. Of this cost \$247.10 was for labor, and \$828.21 for supplies. The distribution of the cost was as follows:

*Distribution of cost of power generated for town of Santa Rita by Chino Copper Co.*

Item.	Per cent.	Cost.	Item.	Per cent.	Cost.
Compressor.....	40	\$430.12	Carpenters' shop.....	1	\$10.75
Machine shop.....	3	32.26	Light service, town lights, etc..	8	86.03
Assaying and sampling.....	1	10.75	Crusher operation.....	26	279.58
Mine drainage; electric pumps.	6	64.52	Total.....	100	1,075.31
Water supply; 3 pumping stations.....	15	161.30			

This cost amounts to 1.02 cents per kilowatt-hour, or 0.76 cent per horsepower-hour. Assuming that the power was used for 720 hours, the number of hours in the month, the cost per horsepower per month was \$5.47.

## WATER SUPPLY AT SANTA RITA.

The somewhat arid climatic conditions of the region give particular importance to a water supply adequate to the needs of such extensive mining operations as those at Santa Rita. However, a series of wells and some small-streams supply ample water and a water-softening plant eliminates the excess of calcium and other salts and renders the supply satisfactory for domestic and boiler use.

The largest well at Santa Rita is the Pinder. It is near the bed of Santa Rita Creek, and much of its supply comes from the sub-surface flow of that creek. A Dean triplex 9-inch by 12-inch electric pump lifts the water from this well and forces it through 3,600 feet of pipe against a difference in elevation of about 410 feet. The diameter of the discharge aperture of the pump and of the first 30 feet of the pipe is 7 inches; the next 2,500 feet is 8-inch pipe, and the last 1,100 feet is 12-inch pipe. The gage on the water pipe near the pump shows a pressure of about 200 pounds persquare inch. Against this pressure the pump throws 175 to 200 gallons per minute, more or less. It is operated by an Allis-Chalmers, 75-horse power, three-phase motor, using a 440-volt current. The curvature of the pipe line and the lengths of the different sizes of pipe are shown in figure 15. The well is also equipped with a steam auxiliary plant for use in case of necessity. During October, 1914, 4,965,230 gallons was pumped from the Pinder well to the water-softening plant, or at the average rate of 10,558 gallons per hour of pumping. In addition, 1,449,782 gallons was pumped for domestic purposes from "drill hole I" and 1,440,450 gallons from the Booth well for general supply.

Each blast-hole churn-drill machine uses about 200 to 300 gallons of water per eight hours. The larger prospecting churn drills each use about 500 gallons per eight hours for all purposes. Steam shovels each require 6,000 gallons, and mine locomotives about 4,000 gallons each per 10 hours.

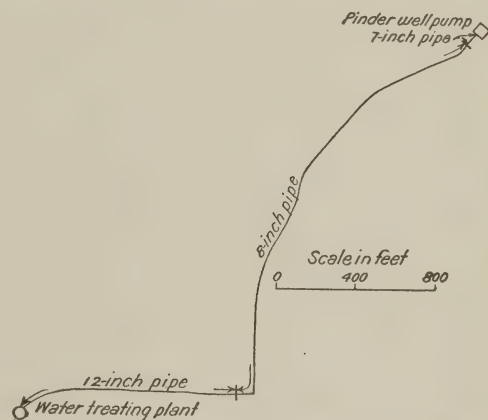


FIGURE 15.—Horizontal curvature and lengths of different sizes of pipe of Pinder well pipe line.

## WATER-SOFTENING PLANT.

A Reisert automatic water-softening plant, with a capacity of 9,000 gallons per hour, takes the excess of calcium and allied salts out of the water so as to make it suitable for boiler use. The plant is situated on a hill, about 300 feet above the town and about 350 feet above the mine, where the water is mostly required.

Figure 16 shows the flow sheet of the plant. The raw water is admitted to the bottom of the saturator, where it comes into contact with cream of lime previously supplied from the lime-mixing or slaking compartment. In rising and mixing with the cream of lime, the raw water reaches the top of the tank in a saturated condition and nearly clear. From the lime saturator it flows into the mixing pan at the top of the settling tank, coming into contact there with raw water and with soda ash solution from the soda compartment. The proper proportions of these inflowing solutions are automatically regulated by float valves. After the raw water, the saturated lime water, and the soda ash solution have been thoroughly mixed they pass from the mixing pan through the downtake pipe to the bottom of the settling tank. From there the mixture slowly rises to the top, and the precipitates resulting from the chemical reaction slowly settle out. At the top the water is fairly clear, but it overflows into filter tanks where the last of the suspended matter is removed. Emerging from these, it goes to a storage tank ready for use. The filter tanks are washed out, whenever necessary, by reversing the flow. The solid matter may be drawn from the bottom of the settling tank by opening a valve.

The plant is operated by one man under the supervision of the mine assayer. The mechanical regulation and operation of the plant can be taught to the average operator in a few days.

## FUEL.

The fuel used by the Chino company comprises bituminous and subbituminous coals from northern New Mexico. Run-of-mine coal and No. 2 lump are principally used. Under normal working conditions about 63,000 tons is used annually at Santa Rita. The average cost of coal delivered there is \$4.35 per ton, more than half of which is for freight charges.

When received the coal is unloaded to storage piles. To keep the temperature down and prevent fires from spontaneous combustion, the piles are built around ventilating stacks, which are wooden conduits a foot square made out of 1-inch or 1½-inch boards in which many holes have been bored (Pl. V, B).



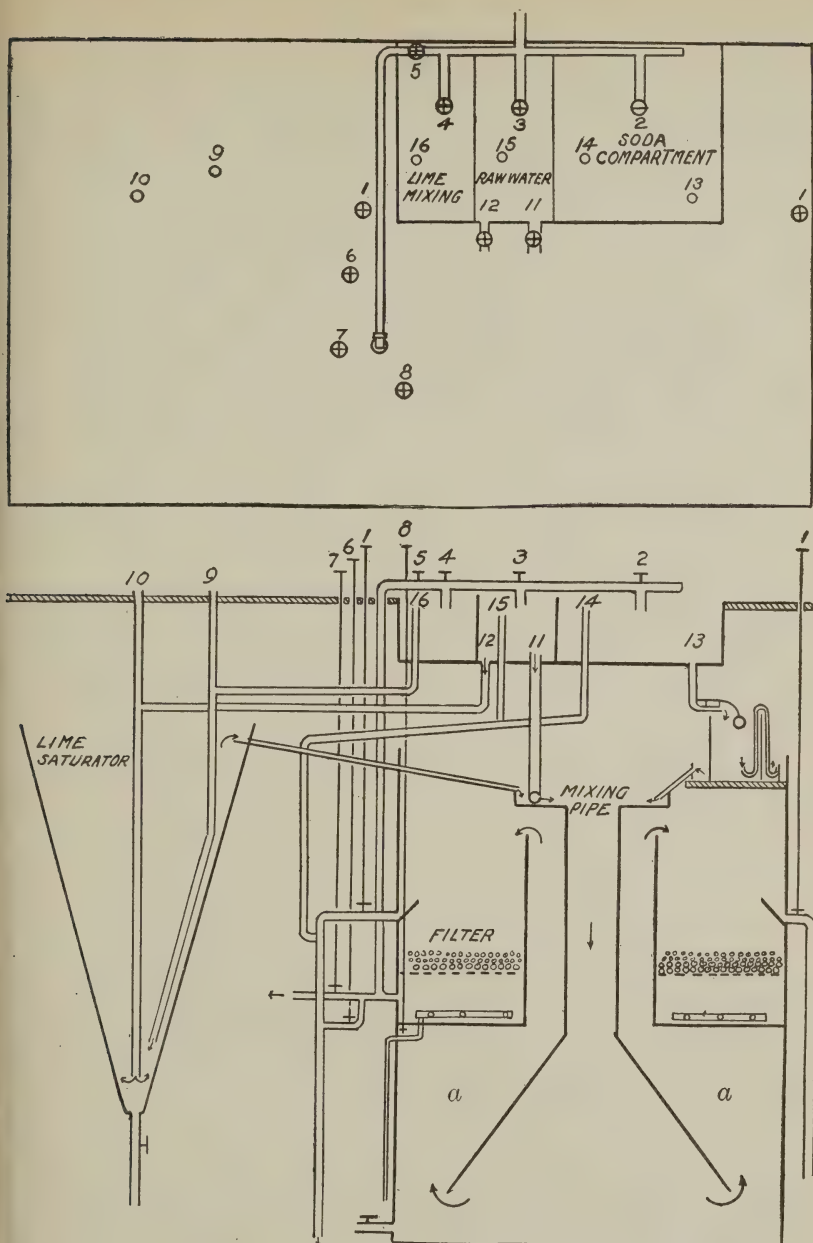


FIGURE 16.—Flow sheet of Santa Rita water-softening plant. 1, Drains filter to top of gutter; 2, raw water inlet to soda compartment; 3, raw-water inlet to raw-water compartment; 4, raw-water inlet to lime-mixing compartment; 5, by-pass to reverse flow of water through filter; 6, drains filter down to tight head; 7, treated-water outlet; 8, air valve for cleaning filter; 9, open pipe for cleaning lime passage; 10, open pipe for cleaning lime-saturator outlet; 11, controls flow of raw water through treating plant; 12, controls flow of saturated lime water; 13, soda-solution outlet; 14, drain for cleaning out soda compartment; 15, drain for cleaning out raw-water compartment; 16, passage from lime-mixing compartment to saturator; aa, settling tank.

### MINING COSTS.

Economy of production is the policy that, other things being equal, offers the greatest opportunity for success in mining, as in most other lines of business. The cost figures established by the Chino company should be of interest, not only to mining men, but also to engineers and contractors who have to do with excavations of various kinds.

The total average cost for blasting, loading, and hauling per ton of ore for the period October, 1910, to September, 1914, was about 20 cents; of this, about 12 cents was for labor and about 8 cents for supplies. The lowest average monthly cost was 12.9 cents per ton, 7.8 cents of which was for labor and 5.1 cents for supplies. The highest average monthly cost was 60.5 cents for July, 1911; 43.5 cents of that cost was for supplies and 17 cents for labor. The purchase of an unusually large store of supplies, charged against this month, caused this abnormally high average.

The total average cost of stripping per cubic yard, from October, 1910, to August, 1914, was 31.5 cents, equivalent to 15.2 cents per ton. Of this, 20.1 cents was for labor and 11.4 cents for supplies. During the same period, the lowest monthly average cost per cubic yard was 23.4 cents, 15.2 cents of which was for labor and 2.8 cents for supplies. The highest monthly average cost was 36.8 cents per cubic yard, of which 24.3 cents was for labor and 12.5 cents for supplies.

During the year 1913, the average cost of mining and hauling to the mill and dumps all classes of material was 36.87 cents per cubic yard. The cost of handling waste alone was 33.43 cents per cubic yard in place, or a little over 16 cents per ton. The cost per ton for blasting, loading, and hauling the ore was 23.13 cents per ton. In general the cost per ton is greater for ore than for waste because of the limited areas in which ore is loaded and the consequently more intermittent character of the loading. Other delays arise from occasional shortage of standard railroad cars for loading ore. The breaking of large blocks of ore by blasting formerly added something to the ore cost. The new plant for crushing coarse material at the mine is intended to obviate such blasting.

### MILLING.

About 10 miles southwest of the terminus of the railroad line at Santa Rita, it emerges from the narrow winding Whitewater Valley and gains the open plateau. Near here, with many miles of open country in which to expand, the mill town of Hurley was laid out, and the concentrating plant was built. The discussion following is not intended to be a complete description of this splendid modern mill, but it calls attention to certain details not mentioned in the account of the mining operations.

The ore is hauled from the mine to the mill by the Santa Fe Railroad Co. Each train, handled by one locomotive and composed of 25 to 27 50-ton steel cars, carries 1,250 to 1,350 tons of ore. At Hurley the cars are run out on a very high trestle over the top of the crude-ore bins, into which they discharge by gravity. The course of the ore through the crushing plant at the mill is shown by figures 17 and 18.

*Description of parts shown in figure 17.*

1. Fifty-ton side-dump ore cars. Weight of car, 44,700 pounds. Length, 35 feet.
2. Grizzlies of 60-pound rails over bins; openings, 12 by 13 inches.
3. Crude-ore bins. Total storage capacity, 14,500 tons; available storage capacity without poking, 5,000 tons; each bin 34 feet wide by 28 feet 6 inches deep by 300 feet long inside.
4. Forty feeders of the caterpillar type. Speed of apron, 10 feet per minute; capacity of each feeder, 1.25 tons per minute—75 tons per hour.
5. Two 30-inch conveyor belts moving 245 feet per minute; about 325 feet from center line to center line.

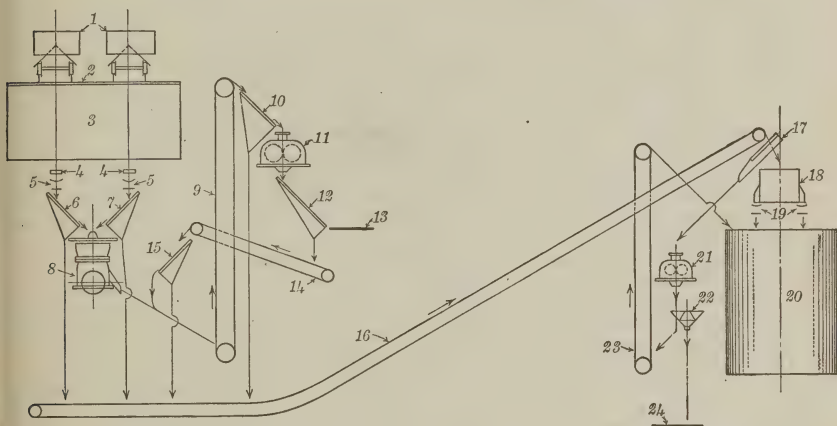


FIGURE 17.—Flow sheet of plant for crushing coarse material, Hurley mill.

6. Grizzly, 1 inch by 4 feet by 12 feet; bars  $\frac{3}{4}$  inch by  $\frac{1}{2}$  inch by 3 inches.
7. Grizzly, 1 inch by 4 feet by 11 feet; bars  $\frac{3}{4}$  inch by  $\frac{1}{2}$  inch by 3 inches.
8. No. 8 McNally crusher; 375 revolutions per minute; crushing to 3 inches maximum size.
9. Elevator; belt 36 inches, 12 ply; head pulley, 60 by 38 inches; boot pulley, 48 by 38 inches; 49 feet 4 inches from center line to center line; belt speed, 440 feet per minute; buckets,  $17\frac{1}{2}$  by 8 by 10 inches; projection, 18 inches between buckets.
10. Grizzly, 1 inch by 4 feet by 9 feet; bars,  $\frac{3}{4}$  inch by  $\frac{1}{2}$  inch by 3 inches.
11. Rolls, 72 by 20 inches; peripheral speed of roll shells when new, 1,000 feet per minute.
12. Grizzly,  $2\frac{1}{2}$  inches by 3 feet by 8 feet 9 inches; for removing large pieces of native copper, wood, etc.
13. Steel platform at bottom of grizzly.
14. Twenty-inch conveyor belt; 20 feet from center line to center line; inclination,  $20^\circ$ ; speed, 330 feet per minute.
15. Grizzly, 1 inch by 3 feet by 7 feet.
16. Conveyor belt, 36 inches wide; about 275 feet from center line to center line; maximum inclination,  $20^\circ$ ; speed, 300 feet per minute.
17. Primary sampler; cuts sample every five minutes.
18. Receiving hopper.
19. Two 24-inch shuttle conveyors; 24-inch belts, 120 feet long; speed, 350 feet per minute.
20. Ten steel tanks, 24 feet 6 inches in diameter by 40 feet high.
21. Rolls, 14 by 27 inches.
22. Vezin sampler, cuts 20 per cent of (21) roll product.
23. Elevator, returning material rejected from samplers to bins for fine ore; 6-ply belt, 6 inches wide.
24. Floor on which sample is quartered by hand.



*Description of parts shown in figure 18.*

1. Fine-ore bins; two cylindrical steel tanks 24 feet 6 inches in diameter by 40 feet deep; approximate capacity 1,000 tons each.
2. Four feeders of the caterpillar type.
3. Two 20-inch belt conveyors, about 20 feet from center line to center line; speed, 250 feet per minute.
4. Two 24-inch wet elevators; 24-inch, 12-ply belts; speed, 400 feet per minute; about 62 feet from center line to center line; buckets staggered on belts; pitch of alternate buckets, 9 inches; buckets 12 by 7 by 8 inches.
5. Six 36-inch by 48-inch impact screens; 600 impacts per minute.
6. Two 42-inch by 16-inch rolls; peripheral speed of shells when new, 1,000 feet per minute.
7. Four 36-inch by 48-inch impact screens; 600 impacts per minute.
8. Two jigs with two 6-mesh screens each; screens 24 by 36 inches; 112 revolutions per minute; 1½-inch stroke.
9. One 30-inch wet elevator; 30-inch, 12-ply belt; speed, 450 feet per minute; 60 feet from center line to center line; buckets, 15 by 8 by 10 inches, staggered on belt; pitch of alternate buckets, 9 inches.
10. Four 4-spigot Richards-Janney classifiers.
11. Two desanding tanks, 5 feet 3 inches by 7 feet 2 inches by 6 feet 9 inches deep.
12. Chilean mill feed tank, 3 feet by 24 feet by 6 feet 6 inches deep.
13. Three Garfield Chilean mills, 32½ revolutions per minute.
14. Two three-spigot Richards-Janney classifiers.
15. Two three-spigot Richards-Janney classifiers.
16. Four five-spigot Richards-Janney classifiers.
17. Six 4-foot by 12-foot Garfield tables; 245 revolutions per minute; ¾-inch stroke.
18. Two 4-foot by 12-foot Garfield tables; 245 revolutions per minute; ¾-inch stroke.
19. Two 4-foot by 12-foot Garfield tables; 245 revolutions per minute; ¾-inch stroke.
20. Two 4-foot by 12-foot Garfield tables; 245 revolutions per minute; ¾-inch stroke.
21. Two 4-foot by 12-foot Garfield tables; 245 revolutions per minute; ¾-inch stroke.
22. Six No. 5 Wilfley tables; 245 revolutions per minute; ¾-inch stroke.
23. One double three-compartment jig; 188 revolutions per minute; 2-inch stroke; 26-inch by 38-inch screen; first and second compartments, 7-mesh, 0.035 brass wire; tail screen, 9-mesh, 0.035 brass wire.
24. Four Isbell vanners; corrugated belts, 6 by 10 feet; 2-inch side stroke; 130 revolutions per minute; 7½-inch slope.
- 25 and 26. Each consists of four vanners same as 24, but having 7-inch slope.
27. Four vanners same as 24, but having 6½-inch slope.
28. Four vanners same as 24, but having 6-inch slope.
29. Four vanners same as 24, but having 7-inch slope.
30. Four vanners same as 24, but having 6-inch slope.
31. Four vanners same as 24, but having 6-inch slope.
32. Twelve vanners same as 24, but having smooth belts; 120 revolutions per minute; 4½-inch slope.
33. Thirty-two vanners same as 32.
34. Twenty-eight vanners same as 32.
- 35, 36, 37, and 38. Each consists of ten steel-cone tanks; 9 feet 5 inches in diameter; 60° slope.
39. Fifteen double wooden tanks; water-recovery plant, equivalent to 50 cone tanks or 10 cone tanks per section; serving five sections.
40. Two slime pumps.
41. Four pumps; 10-inch, special, class "B" volute; two sets of two each, series connected; serve five sections.
42. Two muddy-water sumps, serving five sections.
43. Four pumps; 10-inch discharge; special, class "B" volute, serving five sections.
44. Two sumps for all vanner concentrates, serving five sections.
45. Two 4-inch centrifugal pumps, series connected, taking material from sumps 44, and serving five sections.
46. One 4-compartment Richards-Janney classifier.
47. One 4-foot by 12-foot Garfield table, modified.
48. One 4-foot by 12-foot Garfield table, modified.
49. One 4-foot by 12-foot Garfield table, modified.
- 50, 51, and 52. One vanner same as 24, but having 6-inch slope.
53. Concentrate sump, serving five sections.
54. Three 6-inch centrifugal concentrate pumps, serving five sections.
55. One primary concentrate sampler, Vezin type, serving five sections.
56. One secondary concentrate sampler, Vezin type, serving five sections.
57. Sixteen concentrate bins, 22 feet 6 inches by 15 feet by 12 feet deep. All bins are served by a 4-ton Gantry crane with a 24 cubic foot clamshell bucket.
58. One tailings sampler, pendulum type, cutting the tailings stream once in 6 minutes; serves five sections.

NOTES.—46 to 52 inclusive, constitute concentrate cleaning plant. Three such plants serve five sections.

Note: Cross-hatched machines use muddy water

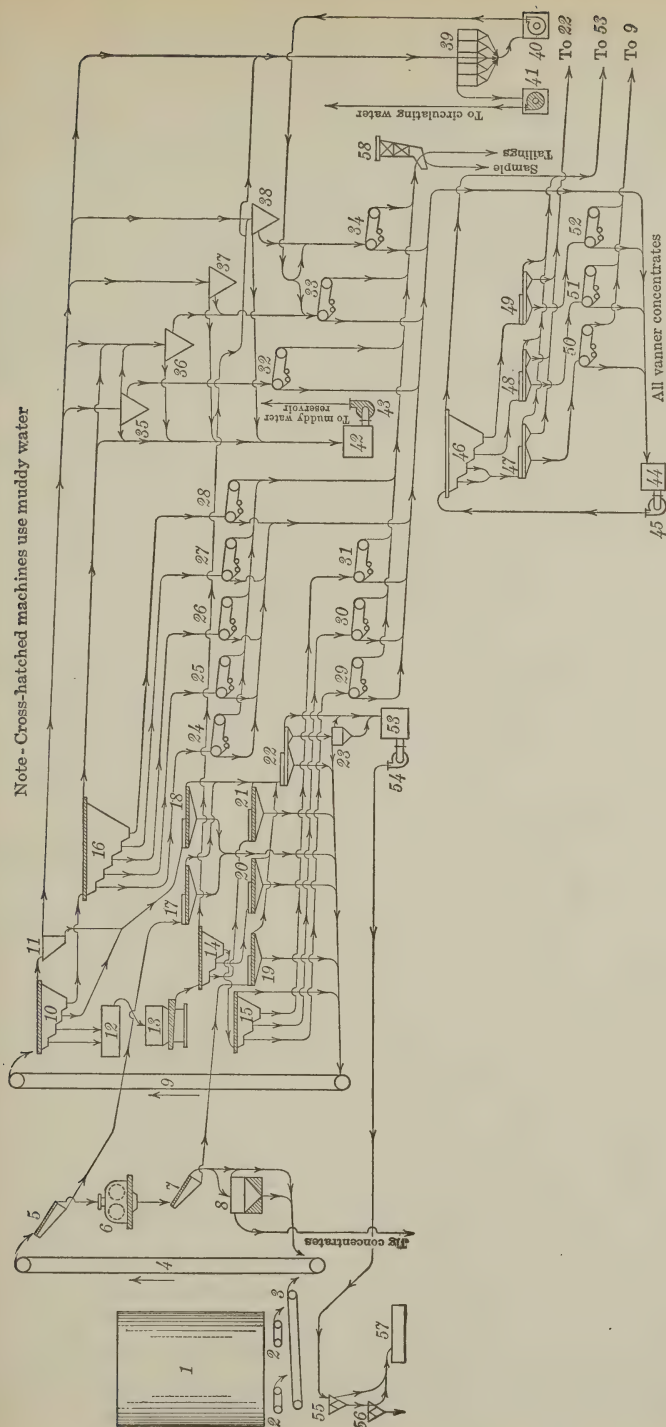


FIGURE 18.—Flow sheet of a section of operations at Hurley mill.

Since the installation of the crushing plant for coarse material at the mine, the grizzlies (2, fig. 17) across the top of the crude-ore bins, consisting of 60-pound rails giving openings 12 by 13 inches, are seldom needed. The capacity of the receiving bins 3 is 5,000 tons, and they discharge, through 40 caterpillar feeders, 4, arranged along the bottom, 20 on a side. These supply two 30-inch conveyor belts, 5, which carry the ore to the second set of grizzlies, 6 and 7 from which the undersizes drop onto the main belt conveyor 16, and the oversizes go into a No. 8 McNally crusher, 8, breaking to a maximum size of 3 inches. From this the total product is raised by an elevator, 9, and passes over a third grizzly, 10, with openings  $\frac{3}{4}$  by  $\frac{1}{2}$  inch by 3 inches. The oversize from this goes to rolls 11 and the undersize to the main belt conveyor 16.

Another grizzly gets the product from the rolls 11. This treatment is mostly to take out, as oversize, flattened pieces of native copper, wood, etc. The undersize goes to another elevator, 14, then over another grizzly, 15. The oversize from this is elevated and again passes through the rolls 11, the undersize going to the main belt conveyor 16. The latter elevates the crushed ore to a receiving hopper, 18, which distributes it onto shuttle-belt conveyors, 19. These discharge into one of 10 steel ore tanks, 24½ feet in diameter and 40 feet high. A primary sampler, 17, cuts a sample from the crushed-ore stream every five minutes. This sample passes through rolls, 21, and is then resampled by a Vezin sampler, 22, the rejected material being returned to the fine-ore bins by an elevator, 23. The sample retained is quartered by hand on a table, 24.

From the fine-ore bins (1, fig. 18) caterpillar feeders, 2, deliver the ore to belt conveyors, 3. From them it is raised by an elevator, 4, and delivered to impact screens 5, with the oversizes going to rolls 6, and the undersizes to Garfield tables 17. The product from the rolls goes to other impact screens 7, the fines from which go to Garfield tables 19, and the oversizes to jigs 8. The jig oversizes go back to the elevator 4, and the concentrates finally reach the concentrate bins. From the Garfield tables the concentrates go to Wilfley tables 22, and the tailings go back by elevator 9 to classifiers 10. Of the five products from these, the two coarsest go to Chilean mills 12 and 13, the finest to desanding tanks 11, the next finest to fine classifiers 16, and the intermediate to Garfield tables 18. All the products from the fine classifiers 16 go to vanners 24 to 28. The products from the Chilean mills 12 and 13 go to classifiers 14, which give three products, the two coarser going to Garfield tables 20 and 21 and the finest to a fine classifier, 15. The three products from the latter go to vanners 29 to 31. The Wilfley tables 22 give three products, the coarse tailings going back by elevator 9 to classifiers 10, the intermediate to jigs 23, and the concentrates to concentrates sump 53 and



thence to storage bins 57. All the concentrates from vanners 24 to 31 go to sumps 44, from which pumps 45 elevate them to the cleaning plant 46 to 52. The tailings from the vanners go out onto the waste dump, being automatically sampled by samplers 58 on the way. The overflow from the desanding tanks 11 and from the classifier 16 goes to a series of steel-cone settling tanks 35 to 38. From these the overflow goes to muddy-water reservoirs 42, where it is somewhat clarified by settling, so as to be used over again. The slimy sediment from the cones goes to smooth-belt vanners 32 to 34, which give tailings that go to the waste dump and concentrates that go to the cleaning plant 46 to 52. This comprises a Janney classifier, modified Garfield tables, and vanners, designed to handle very fine material. The concentrates from the vanners of this cleaning plant are mixed with the concentrates from all the other vanners, and go through the cleaning plant again; the tailings go back to the classifiers 10. The intermediate product from the modified Garfield tables of the cleaning plant goes back to Wilfley tables 22 and the concentrates from the latter go to the concentrates sump 53 and bins, as do the concentrates from the Wilfley tables and the overflow from the classifiers 46 of the cleaning plant.

The foregoing outline, in conjunction with figures 17 and 18, gives in brief form the chief operations of the mill. Since the time of the authors' visit in 1914 alterations and improvements have been added to the plant, but these are perhaps too technical to be treated in any but a report addressed to metallurgists.

The mill has a maximum capacity of almost 8,000 tons per 24 hours, when the character of the ore is favorable, but 6,000 tons is considered a fair daily average. To operate the total milling plant, consisting of five concentrating sections similar to that shown in figure 18, and the coarse-crusher section (fig. 17), about 3,500 horsepower of electrical energy is required.

The average extraction for 1913 was 67.31 per cent from 1,942,700 dry tons milled, equivalent to 5,322.4 tons per day. During a period of about five months a section of the mill was idle owing to modification in the scheme of treatment; hence the tonnage for the year does not represent full capacity. The ratio of concentration for the year was 10.61 to 1, with the concentrates averaging 14.518 per cent copper, or a recovery of 27.37 pounds per ton of ore. The ore treated for the year averaged 2.033 per cent copper. The ratio of concentration was lower than in 1914 because of the high iron content of certain of the ores mined during the year.

The ores treated during 1914 carried less iron and gave a better concentration. For the month of September, 1914, the ratio of concentration was 17.32 to 1, from 101,300 dry tons milled. The lessened tonnage treated was due to the war conditions. The actual recovery

was 73.438 per cent of the copper content of the ore. The concentrates carried 27.67 per cent of copper. Any excess of silica over iron is penalized by the smelter. Toward the end of 1913 a concentrate recleaning plant was added to the mill and this helped to make a cleaner product, thereby lessening the freight and smelter charges on the copper actually produced.

#### HURLEY WATER SUPPLY.

A most important question which had to be solved before a site for the new mill could be decided on was the matter of an adequate water supply. Such a supply, it was found, could be obtained from springs, wells, and the subsurface flow of creeks within piping and pumping distance of the mill site at Hurley. An electrical pumping station on the "B Ranch" has a capacity of 750 gallons per minute. The pump is an 11-inch by 12-inch Aldrich triplex, driven by a 75-horsepower motor, which has to overcome a head of 250 feet. Similar pumping plants are on the Cameron and Whiskey Creeks. The Apache Tejo pumping station has two 14-inch by 18-inch quintuplex Aldrich pumps, each having a capacity of 2,500 gallons per minute; each is driven by a 400-horsepower electric motor, which has to overcome a total head of 400 feet. Only one of the pumps at the Apache Tejo station is operated at a time, the other being held in reserve. The supply of water for domestic use in the town of Hurley comes from this station.

Two reservoirs are in use at Hurley, one for clear and the other for muddy water. The clear-water reservoir gets the water from the pumping stations above described. It is 210 by 144 feet, and 12 feet deep. The muddy-water reservoirs get the settled water, already used in the mill, and caught and partly settled by the Whitewater No. 2 Dam. The size of this reservoir is about 60 by 70 feet and 12 feet deep. The muddy water is handled by six 10-inch, Class B, Worthington volute pumps, each having a capacity of 2,500 gallons per minute against a head of about 95 feet, and each driven by a 75-horsepower motor. The Whitewater No. 2 Dam and the settling reservoirs near it have a storage capacity of about 15,000,000 gallons. The water that flushes the tailings out of the mill is there ponded and settled to be used again. When the new spillway at the Whitewater No. 1 Dam is completed there will be a storage capacity for muddy and flood water of 500,000,000 gallons. The muddy water, after partial clarification by settling, is used over again on certain of the tables and classifiers. It may contain up to 4 or 5 per cent of solids.

The total amount of new water used in the mill during the month of July, 1914, was 52,569,637 gallons. With this about 560,000,000 gallons of muddy water was pumped to be used over again. The

total ore treated during the month was 188,400 tons. The water used per ton of ore treated was 325.6 gallons of new water and 2,961.8 gallons of muddy water, a total of 3,287.4 gallons. The water used for domestic purposes in the town of Hurley during July amounted to 8,768,775 gallons.

#### MILLING COSTS.

The total average cost for all the ore treated in the mill up to September, 1914, was 58 cents per dry ton. In 1914, the lowest average cost for one month was 47 cents per dry ton, for June; during that month 201,000 dry tons was treated. The highest monthly average was 58 cents for the month of September, 1914, with only 101,300 tons milled. Since the mill opened, the total average cost of crushing coarse material (fig. 17) to a maximum size of  $1\frac{1}{4}$  inches, has been 4 cents per dry ton, including supplies, operation repairs, remodeling, etc. During September, 1914, in spite of the lessened tonnage handled, the crushing of coarse material amounted to only 1.8 cents per dry ton. The crushing of fine material to a maximum size of 40 mesh, has cost, on the average, 11 cents per dry ton. For September, 1914, it averaged 9.7 cents. The concentration of the crushed material on the different tables, vanners, jigs, etc., has averaged about 13 cents per dry ton.

The total average milling expense, since the mill opened up to September, 1914, expressed in cents per dry ton of ore treated, is the resultant of the following charges: Light and power, 7.9 cents; water service, 4.8 cents; laboratory and sampling, 1.3 cents; Hurley office expense, 1 cent; engineering and surveying, 0.2 cent; tailings expense, ponding tailings, etc., 0.9 cent; mill heating, 0.1 cent; warehouse expense, 0.5 cent, making a total direct operating expense of 16.7 cents per dry ton of ore treated. A general mill-expense charge of 6 cents, probably covering experimental mill work, etc., together with 6 cents for administration and 1.4 cents for maintenance, raises the cost of mill-operation to 30.1 cents. This, when added to the cost of crushing coarse and fine material and concentrating, 4, 11, and 13 cents, respectively, makes the grand total of 58.1 cents per dry ton treated.

When the mill again begins to handle ore up to its full capacity, the recent adjustments and improvements made in it should considerably better these general average costs.

In connection with the Hurley plant there is a large assay shop where about 100 assays per day are made. Experimental work on the ores is also carried on in a well-equipped laboratory, with the object of still further reducing the cost and increasing the efficiency of concentration.



## REPAIR SHOPS AT HURLEY.

At Hurley there are also storehouses and extensive machine and repair shops. The completeness of the latter is indicated by the following list of equipment:

*Equipment in Chino company shops at Hurley.*

## Equipment in mechanical department.

- One Woodward & Powell, jr., patented 42-inch by 42-inch by 12-foot planer.
- One J. G. Blount No. 5 double-head emery grinder.
- One New Haven 50-inch by 24-foot triple-gear, screw-cutting engine lathe.
- One Prentiss 24-inch by 22-foot engine lathe.
- One Davis 12-inch by 8-foot quick-change gear lathe.
- One Steptoe 20-inch back-gear, crank shaper complete.
- One Blount 20-inch wet tool grinder, complete, with 20-inch by 2½-inch emery wheels.
- One Yankee twist-drill grinder.
- One Williams No. 5 12-inch pipe machine.
- One Niles 5-foot semiuniversal radial drill.
- One Hofer 21-inch vertical drill press.
- One Owen No. 2 milling machine.
- One Acme 2-inch single-end bolt-threading and nut-tapping machine, class "A."
- One Cleveland Punch and Shear Co. No. 0 plate-bending rolls.
- One Robertson No. 2 rapid power hack saw.
- One Ryerson 1,200 pound single-frame steam hammer.
- One American 36-inch lathe, with 18-inch bed, double back-gear.
- One American No. 3 high-speed radial drill.
- One Higley No. 12 cold metal saw.
- One Ryerson-type, radial drill, with 8-foot arm.
- One Ryerson & Son Cleveland universal splitting shear.
- One American Tool Works 18-inch, new-pattern, heavy-duty engine lathe.
- One Cleveland Punch & Shear Co., style C, solid-frame single end punch-and-shearing machine.
- One Acme 1-inch single-bolt cutter, class "A."
- One Green River No. 55 opening die bolt cutter.
- One Racine high-speed power hack saw, 6-inch capacity.
- One Brown & Sharpe cutter grinding machine No. 2.
- One Lodge & Shipley screw-cutting heavy-duty engine lathe.
- One Racine high-speed draw-cut hack saw, 6-inch capacity.

## Carpenter-shop equipment.

- One power mortising machine.
- One 24-inch single-cylinder planer and matcher.
- One 24-inch hand planer and jointer.
- One 16-inch variety saw.
- One Universal 16-inch woodworker's saw.
- One single-spindle vertical boring machine.
- One 36-inch band saw.
- One pattern maker's wood lathe, 24-inch swing.
- One Buffalo knife grinder.
- One Universal No. 8-f trimmer.
- One double emery-wheel stand.
- One 36-inch grindstone with iron frame.
- One 24-inch rip saw.
- One cut-off saw, 36-inch swing.

During the month of June, 1914, 33 mechanics and helpers were employed in the mechanical department, and 8 carpenters and helpers in the carpenter shop of the Hurley shops.

During the month of June the total cost of repairs at Hurley, including labor, supplies, and overhead charges, was 1.96 cents per dry ton of ore milled. The cost for September, 1914, was 1.89 cents, and for April, 1915, 1.32 cents per dry ton milled.

### GENERAL ENGINEERING DETAILS.

Where operations are so extensive as those at Santa Rita, necessarily a wide range of engineering problems is presented for solution. Not only is much field work and mapping necessary, but a great deal of yardage measuring and computing must also be done each month. Railroad-engineering problems are common, and jobs in machine design, architecture, reinforced concrete work, hydraulics, etc., make calls on the engineering staff from time to time. This staff embraces a chief engineer and five to eight assistants.

### MONTHLY ESTIMATES AND STATEMENTS.

At the end of every month stripped and mined areas are measured, and the yards and tonnage are computed. The measurements are made by stadia, and the information is plotted on a coordinated and contoured working map, which has a scale of 30 feet to the inch. The stadia stations are accurately located by transit from triangulation points. A field party consists of a transitman, a recorder, and two rodmen. The field notes are recorded in the transit book in the following form:

*Specimen record in transit book.*

Date .....

Low Line No. 8 Dump.

Party: ....., Transitman; ....., Recorder; ....., Rodman (top).

....., Rodman (bottom).

[Field page.]

[Office page.]

Station. <sup>a</sup>	B. S. <sup>b</sup>	Horizontal angle. <sup>c</sup>	Stadia distance. <sup>d</sup>	Vertical angle. <sup>e</sup>	High or low sights. <sup>f</sup>	Horizontal distance. <sup>g</sup>	Vertical distance. <sup>g</sup>	Elevation. <sup>h</sup>
	Δ N.							
12 (H. I. 4.9); elevation.		352°00'	408	+3°26'		408	+24.5	6370.4
1,345.9.....		355°15'	388	+4°45'	+2ft. (B.L.)	386	+32.1	6376.0
		358°15'	370	+3°43'		370	+24.0	6369.9

<sup>a</sup> Transit station.

<sup>b</sup> Backsight (on triangulation station N).

<sup>c</sup> Read to nearest 15 minutes.

<sup>d</sup> Stadia distance read to the nearest foot.

<sup>e</sup> Read to nearest minute.

<sup>f</sup> High or low sights read (called "boot leg").

<sup>g</sup> Actual, obtained from stadia reduction tables.

<sup>h</sup> True elevation of ground calculated.

When information similar to that contained in the specimen statement is put on the map, a color distinctive from that used the previous month is employed. Sections across the entire cut or dump are taken every 25 feet and are plotted as shown in figures 19 and 20.

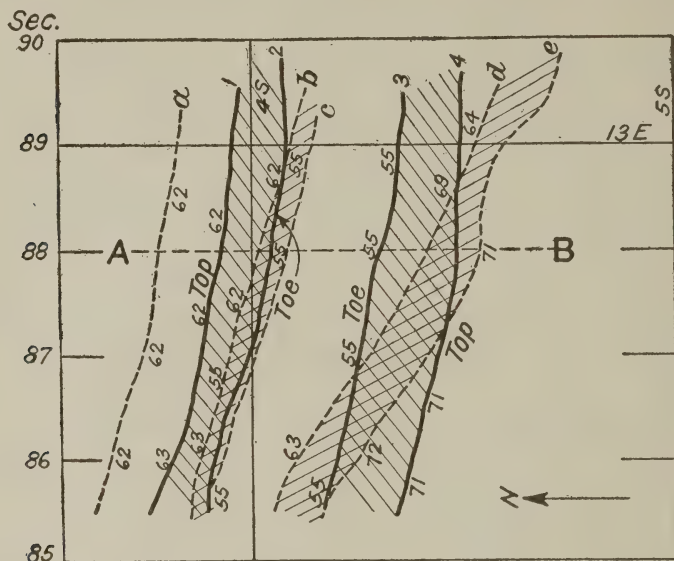


FIGURE 19.—Plan map of excavation and of filling during one month. The dotted lines *a* to *e* are each drawn through points having approximately the same elevation, on surface of previous month's excavation; solid lines 1 to 4 show points having same elevation on present surface. Elevation is indicated in feet. The figures 85 to 90 refer to sections on plotting map; 5S, 13E, refer to coordinates of drill holes shown on engineer's records of yardage of excavation.

The surface as measured the previous month is shown on the cross section (fig. 20) by dotted lines, and the more recent surface is shown by solid lines. A plan map of excavation and filling for one month

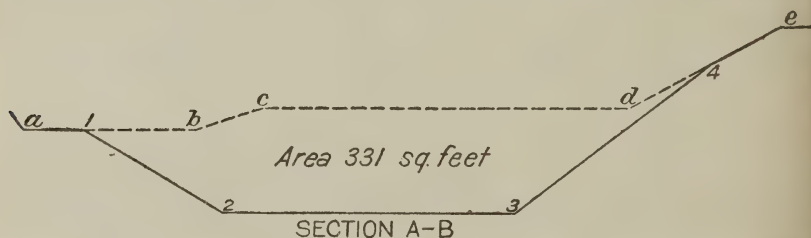


FIGURE 20.—Plotted cross section of month's excavation. Area represented is that of a vertical cross section through A B, in figure 19. The dotted line represents surface as measured in previous month.

is shown in figure 19, which represents an area of which figure 20 shows a cross section.

The end areas are scaled directly off the cross sections, and the average of the two end areas, multiplied by the distance between the cross sections, gives the cubical contents of each excavated block.



The plotting and computations are checked, and then the results are entered in the "Steam-Shovel Computation Book" in the following manner:

*Specimen entries in "Steam-shovel Computation Book."*

Section No.	Total cut. <sup>a</sup>		Ore estimate.		
	Area.	Volume.	Area.	Average end area.	Volume.
(Wedge, 15 feet) to 95.....	<i>Sq.ft.</i> (Noted if any.)	<i>Cu.ft.</i> (Noted if any.)	<i>Sq.ft.</i> 14	<i>Sq.ft.</i> 7	<i>Cu.ft.</i> 105
95.....			14		
94.....			56		
			70	35	875
94.....			56		
93.....			174		
			230	115	2,875
93.....			174		
92.....			284		
			458	229	5,725

<sup>a</sup> Stripping and mining.

Under the column headed "Section No." is recorded the wedge of ground excavated during the preceding month between cross sections 96 and 95. Under "Total cut" is shown the total excavation of waste and ore; under "Area" the areas of the cross sections in square feet are given, cross section 95 had an area of 14 square feet, and cross section 94 one of 56 square feet. The average area of the two sections, 35 square feet, is given under the column designated "Average end area." Multiplying this by 25 feet, the distance between the two cross sections, gives 875 cubic feet, the volume of the prism between sections 94 and 95, removed during the month. In like manner the cubical contents of all the other prisms are calculated. In the calculations 13 cubic feet of rock in place, and 21 cubic feet of broken rock, are used as the equivalent of a ton of 2,000 pounds. Additions to and subtractions from ore and waste dumps are calculated in the same manner.

The records of excavation for each bench are reported separately, as are the records of the excavation work of each steam shovel. If a shovel excavates from two or more benches during the month, the proportion moved from each is determined from the bench and the shovel records, and the proper apportionment made. The assay records and the number of cars sent to the crusher, direct to the mill, or direct to the smelter, determine the amount of the material classed as ore. The number of cars sent to the waste dumps determines the amount of the material excavated classed as waste. Of course, these data are checked against the engineers' estimates of solid cubic yards moved, etc.

A monthly comparative statement of the work done by each shovel is compiled for each month. This table also shows the number of cars of material loaded by each steam shovel and the capacity of each car. Records are kept by each steam-shovel foreman of the number and capacity of cars loaded and whether they contain ore or waste. The table following shows the results of steam-shovel operations during July, 1914:

*Results of steam-shovel operations at Santa Rita during July, 1914.*

Steam shovel No.	Solid cubic yards of waste handled. <sup>a</sup>	Tons of ore handled. <sup>b</sup>	Material in place handled, solid cubic yards. <sup>a</sup>	Shifts worked. <sup>c</sup>	Average solid yards per day (two shifts).
1.....	14,222	40,415	33,681	50.5	1,334
2.....	57,437	-----	57,437	56.2	2,485
3.....	44,243	26,665	57,082	56.0	2,482
4.....	74,469	-----	74,469	55.9	2,664
5.....	57,648	2,451	58,828	49.7	2,367
6.....	28,953	8,909	33,242	45.5	1,461
7.....	20,590	38,305	39,033	56.8	1,668
8.....	47,356	1,466	48,062	51.6	1,863
9.....	11,691	108,010	63,696	57.1	2,231
10.....	-----	4,862	2,341	11.2	418
Total.....	356,609	231,083	467,871	488.5	1,916

<sup>a</sup> Compiled from engineers' cross-section estimates.

<sup>b</sup> Taken from mill returns.

<sup>c</sup> From steam-shovel foreman's report.

A table showing the broken-rock yardage capacity of all the different ore cars used is a convenience in making estimates. It shows, for instance, that a car having a capacity of 100,000 pounds can carry 38 cubic yards; a car having a capacity of 80,000 pounds is loaded with 30 cubic yards; a car having a capacity of 60,000 pounds carries 23 cubic yards; and a car of 55,000-pound capacity contains 21 cubic yards of broken rock. For convenience of calculation this table is extended so that one can see at a glance that a train of, say, 50 cars of 100,000-pound capacity can carry 1,900 cubic yards of ore.

The engineering department also posts a special form that serves as a daily indicator of steam-shovel operations for the information of the manager and superintendents. A specimen form follows:

*Specimen report of day's steam-shovel operations, for information of mine manager and superintendent.*

Date.	Shift (D=day N=night).	Accumulative car yards per shift. <sup>a</sup>	Total car yards per day. <sup>a</sup>	Accumulative car yards per day. <sup>a</sup>	Accumulative for month. <sup>b</sup>	Comparison with previous month. <sup>c</sup>	Remarks.
8.....	D.....	480	480	480	622	x 142	Shovel No. — shut down for repairs.
9.....	D.....	1,056	576	1,056	622	+ 434	

<sup>a</sup> From individual shovel records.

<sup>b</sup> From previous month's record.

<sup>c</sup> Shows gain or loss as indicated by difference between figures in two preceding columns. x indicates car-yard shortage, as compared with the previous month; + indicates car-yard increase.

The engineering computations in the office are greatly facilitated by detailed and comprehensive tables and diagrams, prepared for reducing the stadia observations, giving elevations, horizontal distances, etc.

On a plain coordinated wooden surface, about  $4\frac{1}{2}$  by  $6\frac{1}{2}$  feet, the drill holes are accurately located, and each is shown as a  $\frac{1}{8}$ -inch hole bored into the wood. In these holes  $\frac{1}{8}$ -inch round sticks are placed. The height of the stick above the surface represents the depth of the hole. On each stick, the ore body or bodies that the hole cuts are painted, according to scale, showing the proper thickness and spacing. This gave a fairly good chart, or model, of the ore bodies, showing their depth, thickness, etc. However, this chart proved much less useful than the cross sections of the ore bodies prepared from the drill-hole data.

### MONTHLY, QUARTERLY, AND ANNUAL STATEMENTS BY ENGINEERING DEPARTMENT.

Monthly, quarterly, and annual statements and tables are compiled by the engineering department to give the following information: The amount of ore excavated, milled, and smelted; the total waste excavated and hauled to the dumps; the yardage of ore and waste taken from each bench; the costs of the various operations; the amount of ore remaining; its content of copper and value under given conditions, and other such data. In order to illustrate the manner in which these data are tabulated the following self-explanatory forms are given:

#### ORE AND STRIPPING.

Steam-shovel mining ..... ore body.

SANTA RITA, N. MEX., .....

For month of .....

Bench.	Solid rock, cubic yards in place.	Earth and dumps, cubic yards in place.	Total cubic yards.	Cubic yards of stripping.	Per cent of copper in stripping.	Tons of ore.	Per cent of copper in ore.	Pounds of copper.

Remarks:

[This form is amplified to contain accumulative records of totals to date from Jan. 1.]



**STEAM-SHOVEL OPERATION.**

SANTA RITA, N. MEX., .....

For ..... quarter, 19....

Steam shovel No.	Cubic yards of waste handled.	Tons of ore handled.	Yards in place handled.	Shifts worked.	Average solid yards per day (twoshifts).

Remarks: .....

**TOTAL STEAM-SHOVEL WORK.**

SANTA RITA, N. MEX., .....

For month of .....

Ore body.	Solid rock, cubic yards in place.	Earth and dumps, cubic yards in place.	Total cubic yards.	Cubic yards of stripping.	Per cent of copper in stripping.	Tons of ore.	Per cent of copper in ore.	Pounds of copper.

Remarks: .....

[This form is amplified to contain accumulative records of totals to date from Jan. 1. A form with similar headings is used for quarterly reports.]

**WASTE DUMPS.**

SANTA RITA, N. MEX., .....

For month of .....

Dump.	Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.	Cars.	Tons per car.	Cubic yards per car.	Totals to date from January 1.			
								Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.

Remarks: .....

**ORE PILE.**

SANTA RITA, N. MEX., .....

For month of .....

Pile.	Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.	Cars.	Tons per car.	Cubic yards per car.	Totals to date from January 1.			
								Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.

Remarks: .....

**MILL SHIPMENTS.**

SANTA RITA, N. MEX., .....

For month of .....

Ore from—	Shipping weight, tons.	Per cent of moisture.	Dry weight, tons.	Per cent of copper.	Pounds of copper.	Cars.	Tons per car.	Totals to date from January 1.				
								Shipping weight, tons.	Per cent of moisture.	Dry weight, tons.	Per cent of copper.	Pounds of copper

Remarks: .....

**SMELTER SHIPMENTS.**

SANTA RITA, N. MEX., .....

For month of .....

Ore from—	Shipping weight, tons.	Per cent of moisture.	Dry weight, tons.	Per cent of copper.	Pounds of copper.	Cars.	Tons per car.	Totals to date from January 1.				
								Shipping weight, tons.	Per cent of moisture.	Dry weight, tons.	Per cent of copper.	Pounds of copper

Remarks: .....

## ORE AND WASTE DUMPS.

## MILL SHIPMENTS.

SANTA RITA, N. MEX., .....

For ..... quarter, 19...

Ore dumps.					Waste dumps.				
Dump.	Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.	Dump.	Cubic yards.	Tons.	Per cent of copper.	Pounds of copper.

## SMELTER SHIPMENTS.

Ore from—	Shipping weight, tons.	Per cent of moisture.	Dry weight, tons.	Per cent of copper.	Pounds of copper.	Cars.	Tons per car.

Remarks:

.....

## HEALTH AND SANITARY MEASURES.

## HOSPITAL AT SANTA RITA.

When the new company acquired the Santa Rita property and began operations on an extensive scale, a suitable hospital for their employees seemed highly desirable. Consequently, a well-ventilated one-story frame structure with wide open porches was built and adequately equipped. The hospital has been in existence four years, and in that time with a staff of two doctors and four nurses, it has successfully treated 1,314 cases. It is maintained by fees of \$1.50 per month from all single employees and \$1.75 from all married employees. The fees entitle the employees and their dependents to hospital services, medicine, and treatment during illness. Separate charges are made, however, for childbirth cases and also for cases of illness resulting from drunkenness or fighting, and for treatment of venereal diseases. Patients not entitled to free treatment give \$2 a day in addition to nominal charges for surgical operations.

The following data regarding cases treated during the two years ended February 15, 1913, show how useful and successful the hospital has been:



*Data regarding cases treated in company hospital during two years ended Feb. 15, 1913.*

Cases in hospital.....	665
Deaths.....	21
General operations.....	106
Abdominal operations.....	32
Deaths following operations.....	<sup>a</sup> 2
Pneumonia cases.....	34
Deaths from pneumonia.....	5
Accident cases.....	157
Deaths from accident.....	2
Typhoid cases.....	37
Deaths from typhoid.....	1
Obstetrical cases.....	35
Deaths from obstetrical cases.....	0

## CAUSES OF DEATHS.

Tuberculosis.....	2
Organic heart disease.....	1
Pneumonia.....	5
Cancer.....	1
Enterocolitis.....	2
Hookworm.....	1
Following injuries.....	2
Meningitis.....	1
Typhoid.....	1
Cerebral hemorrhage.....	2
Gunshot wound.....	1
Peritonitis.....	1
Syphilis.....	1
Total.....	<sup>b</sup> 21

## ACCIDENT REPORTS.

When an accident occurs to an employee his foreman at once issues a report on the following form to the general office:

## NOTICE OF AN ACCIDENT TO AN EMPLOYEE.

Name of person making report.....	Occupation.....
.....Street.....	.....City.....State.....
Date and hour of accident.....	191.., ..m.
Date of this report.....	191...
Injured person:	
Name.....	Address.....
Age.....	Family..... Weekly wages, \$.....
Occupation.....	
How long employed in this work?.....	
How long employed prior to accident?.....	
General duties.....	

<sup>a</sup> One was advanced appendicitis and one was gunshot wound of stomach, liver, and intestines.

<sup>b</sup> Of these, 7 did not live 2 hours after admission to hospital.

## The injury:

Nature and extent.....  
 Was surgical aid rendered?..... By whom?.....  
 Taken home or to hospital?..... Probable length of disability.....  
 Has injured returned to work?.....

## The machine or appliance causing the accident:

What was it?.....  
 State condition..... When last inspected?.....  
 Whose control at time of accident?.....  
 Was there any defect in the machine or appliance? State fully:.....  
 Was light good?..... Have broken parts (if any) been preserved?.....

## The accident:

Place?.....  
 Was accident due to carelessness of injured or negligence of fellow workmen?  
 If so, of whom?.....  
 Statement of injured.....  
 Name and address of foreman in charge of work.....  
 Names and addresses of witnesses: .....

## DESCRIPTION OF THE ACCIDENT.

(If necessary, to show cause of accident, draw rough sketch on back hereof.)

[Space for description.]

This notice made out by: .....

(State occupation.)

When the injured person has recovered, the following form certificate is executed:

*Form for certificate of injury.*

Claim No. ....

## LIABILITY DEPARTMENT.

## SURGEON'S REPORT AND CERTIFICATE OF CLAIM.

1. Name of injured..... Occupation.....
2. When first seen after accident.....
3. Precise nature, location, extent of injuries and how received.....
4. Were such injuries the direct result of said accident.....
5. I attended claimant from ..... to ....., 19...., during which time the injuries above described constituted the sole and only cause of disablement; and I certify that he was totally incapable of following his usual occupation, because of such injuries, from the ..... day of ....., 19...., to and including the ..... day of ....., 19....

Surgeon.

## CERTIFICATE OF EMPLOYER, SUPERINTENDENT, OR TIMEKEEPER.

This is to certify that ..... herein referred to, is insured under policy ..... and was in my employment at a remuneration of ..... per hour; that he was injured at the time and in the manner stated and, in consequence thereof, sustained continuous and total loss of time for the period of ..... hours, amounting to \$.....

Date .....

(Position.)

## RELEASE.

Received of ..... the sum of .....  
 (\$. ....) dollars, in full satisfaction and discharge of any and all claims  
 I have or may have against said ..... because of a  
 certain injury or injuries resulting from an accident sustained by me on or about  
 the ..... day of ....., 19...., while in the  
 employ of the said ....., and further for any benefits that  
 may be accrued to me by reason of said accident.

(Claimant.)

Date .....

Witness:

The number and the nature of accidents at the Santa Rita mine  
 and at the Hurley plant for the years 1912 and 1913 are given in the  
 following table:

*Data regarding accidents at Santa Rita mine and Hurley plant, 1912, 1913.*

	Year.	Killed.	Per cent.	Seriously injured.	Per cent.	Slightly injured.	Per cent.	Average number of em- ployees per day worked.
Santa Rita mine.....	1912	3	0.05	20	0.30	117	1.75	553
	1913	12	.11	25	.24	234	2.21	880
Hurley mill.....	1912	3	.036	15	.20	123	1.50	697
	1913	1	.01	5	.05	104	1.09	797

## SANITARY EQUIPMENT.

A large number of the new company's houses are provided with modern plumbing. The sewage from these houses is discharged into cesspools. The smaller houses of the laborers are on large sloping lots and the waste water from the kitchens, baths, etc., is run out into the sand and gravel of the sloping surface and absorbed. Privies for these houses are placed at the rear of spacious lots, usually 100 feet away from the house. They are fitted with metal receptacles, which are removed periodically, and the contents thrown on the waste dumps and covered by carloads of waste. As a part of his general duties, the company's peace officer under the supervision of the chief physician maintains a vigilant watch over the sanitary condition of the camp.

## RESCUE AND FIRST-AID WORK.

The Santa Rita main office has provided a suitable locker section in which three Draeger apparatus are stored. The equipment also includes duplicate oxygen bottles with water gage and liter bag. Hand flash lights are used exclusively.



Five employees of the company have been trained by Bureau of Mines field men, and received the bureau's certificates during the early spring of 1914; these five men have had periodic practice drills. It is the intention to train at least 10 more men in the use of the breathing apparatus.

The headquarters of the camp physician and his assistant is the company hospital, and on account of its close proximity to the mining operations no first-aid teams have been organized. The physician delivers periodic lectures on sanitation and elementary first-aid methods.

#### RECREATION.

The company officials encourage and contribute to the support of suitable forms of amusement for the employees and their families. Baseball and basketball teams are organized and participate in games arranged with similar teams from Hurley and Silver City. A rifle team enjoys the privileges of such teams as come up to the standards in the United States Army Regulations, obtaining healthful and instructive diversion. Two or three annual picnics give delightful outings to the men, women, and children of the camp.

#### TIME KEEPING, ACCOUNTING, AND WAREHOUSE METHODS.

A good working organization is one of the fundamentals of success in any business. A few years ago this factor was largely ignored by mining companies, but to-day it finds place in most of those that are successful. It, therefore, seems well worth while to call attention, in a general way, to the efficient system of time keeping, accounting, and warehouse methods in use at Santa Rita.

#### TIME KEEPING.

When skilled laborers are employed by the company, they are asked to fill out Form 10 so that some idea may be had of their responsibility and past achievements. From an unskilled laborer, no references are required other than a statement as to his last employment. Each foreman, on hiring any one, turns into the office a slip (Form 11). If the person employed is a skilled laborer, this form is yellow in color, and has the notice: "Bearer to furnish references." Each employee is given a number and is furnished with a cardboard check, 1½ by 2 inches, bearing that number.

FORM 10.—Form filled in by applicant for position as skilled laborer.<sup>a</sup>

## APPLICATION.

Date.....191...

MINING Co.,

*Santa Rita, New Mexico.*

GENTLEMEN: I hereby make application for work as.....

and refer you to ..... for my previous

(Give name of company and foreman, also address.)

record, and I hereby authorize and request ..... to furnish same,

including the cause of leaving ..... service, if known. And I hereby release

said ..... from all liability for any

damage whatsoever on account of furnishing such record.

Witness: .....

The above applicant ..... was employed

by the undersigned as ..... from .....

to .....

Cause of leaving .....

Ability .....

Habits .....

Services were .....satisfactory.

Remarks: .....

Dated: ..... Signed: .....

FORM 11.—Form used in notifying timekeeper of the hiring of a new employee.<sup>b</sup>

.....191...

TO TIMEKEEPER:

I have hired .....

as a ..... at .....

per day. ....

*Foreman.*

At the time office, which is conveniently situated, each employee, except the office force, the locomotive engineers, the firemen, and the steam-shovel crews, check in at the beginning of every shift, by calling out his number to the clerk at the window of the time office. The clerk puts opposite each man's number a horizontal black mark if the man is on the day shift and a red if on the night shift. The numbers are arranged on a daily pay-roll balance sheet, a section of which is shown as Form 12. The next check on the working time of the employees is that obtained by the time inspector, who goes around checking up the men found on the work. Forms 13a, 13b, and 13c indicate the character of the records kept in the time inspector's note book. These are left at the timekeeper's office to be checked with a perpendicular mark against each man's number on the daily pay-roll balance sheet. After the morning horizontal mark and the time inspector's perpendicular mark, each workman checked in has a plus (+) sign after his name on the sheet.

<sup>a</sup> Size of form used by operating company, 8½ by 11 inches.<sup>b</sup> Size of original form 3½ by 4½ inches.

FORM 12.—Register kept at time office.<sup>a</sup>

## DAILY PAY-ROLL BALANCE SHEET.

[Check men on shift with horizontal line in check column. Use black ink for day shift and red ink for night shift. Cross check from timekeeper's taken on the works. Enter hours and amount from work slip turned in at the end of each shift.]

0	Check.	Hours.	Amount.		1	Check.	Hours.	Amount.	
1									
2									
3									
4									
5									
6									
7									
8									
9									
10									

<sup>a</sup> Form used by operating company, 19½ inches high by 36½ inches wide. Provision made for 1,500 names on a sheet, 100 names in a column. Horizontal blue lines separate each number, every tenth number being followed by a purple line.

FORM 13a.—Form used by time inspector for recording "time" of miscellaneous employees.<sup>a</sup>

## MISCELLANEOUS LABOR.

....., 191..

Day Night	No.	Hrs.	Remarks.
Car trimmers			
Switch tenders			
Switch tenders			
Flagman			
Water boys			

<sup>a</sup> Size of original form, 4½ inches wide by 7½ inches high, with three punch holes on left margin for binding.

FORM 13b.—Form used by time inspector for recording "time" of steam-shovel crews.<sup>a</sup>

SHOVEL NO. .... Ore.....Hrs.  
Strip.....Hrs. ...., 191..

Day Night	No.	Hrs.	No.	Hrs.
Engineer				
Fireman				
Craneman				
Coalman				
Pitmen				
Dumpmen				

<sup>a</sup> Size of original form, 4½ inches wide by 7½ inches high, with three punch holes on left margin for binding.



FORM 13c.—Form used by time inspector for recording "time" of locomotive crews.<sup>a</sup>

....., 191..

## ACCOUNT:

		No.	Hrs.	No.	Hrs.
Foreman	b {				
Eng. No.					
Eng. No.					

<sup>a</sup> Size of original form, 4½ inches wide by 7½ inches high, with three punch holes on left margin for binding. Original form provides for record of 11 engines instead of 2, as shown.

<sup>b</sup> E, engineer; F, fireman; B, brakeman.

For the men engaged on special jobs that are to be charged to certain accounts, and also as an additional check on working time, Form 14 is used. This form is issued in different colors for the different occupations, and the occupation and the rate of pay of the employee are entered on each. For special jobs by skilled laborers, chargeable to certain accounts, Form 15 is used. If a special gang is engaged on work chargeable to different accounts, the distribution of the charge against each account is entered on Form 16.

All these forms, by keeping track of the proper distribution of charges, greatly facilitate bookkeeping. Each man's time record is kept in a large loose-leaf book composed of pages similar to that indicated by Form 17. At the end of the month this is turned in to the main office.

When an employee is discharged or quits the company's service, he is given a slip (Form 18) by his foreman. On the back of this his store, board, and rent bills are accounted for, or marked "none." The indorsements are made in ink and are signed by the officers responsible for them. The timekeeper then fills out Form 19, puts it in a sealed envelope, and hands it to the employee, who takes it to the cashier and receives his pay.

In common with most large employers, the mining company keeps personal record cards for each employee; the scope of these is shown by Form 20.

FORM 14.—Form used in recording "time" of men employed on special jobs, the expense of which is to be charged to certain accounts.<sup>a</sup>

Occupation	Rate		
.....	.....		
Date.....			
P. R.	Hrs.		
No. ....	Worked.....	Amount \$ .....	
Charge to .....			
.....			
.....			
Foreman.....			

PUNCH HERE HOURS WORKED

1	14
2	15
3	16
4	17
5	18
6	19
7	20
8	21
9	22
10	23
11	24
12	25
13	26

FORM 15.—Form used in connection with special jobs by skilled laborers, chargeable to certain accounts.<sup>b</sup>

## CARPENTERS.

SANTA RITA MINE,

Name..... Date.....191.....  
Check No.....

Give full description of work done.	Hours.	Rate.	Amount.
Total			

Examined and approved:

Foreman.

a Size of original form, 3 by 5 inches.

*b* Size of original form, 4½ inches high by 7 inches long, on heavy flexible stock.

FORM 16.—Form used in connection with work done by a special gang and chargeable against each of several accounts.<sup>a</sup>

SANTA RITA MINE,  
....., 191 ..

Occupation.	Number.	Hours.	Rate.		Amount.	

Charge to:

Examined and approved:

Foreman.

FORM 17.—Form used in time office for keeping each man's time for a month.<sup>b</sup>

### TIME RECORD.

Name..... No.....

Occupation.	1	2	3	29	30	31	Total time.	Rate.	Amount.

Remarks.								Total.
Deductions.	Doctors and hospital							\$
	Insurance							\$
	Store							\$
	Board							\$
	Rent							\$
Check No. ....								Bal. due ..

FORM 18.—Slip given by foreman to employee who is discharged or quits.<sup>c</sup>

SANTA RITA, N. MEX., ..... , 191 ..

TIMEKEEPER:

Give bearer.....

(No. ....) his time for services to date.

Quit } Mark  
Discharged } with  
No work } an X.

Signed.....

### IMPORTANT.

Get this cleared at Santa Rita store, Santa Rita boarding house, and bunk house, then present at the office for payment between the hours of 1 p. m. and 3 p. m. daily except Sunday.

(Over.)

<sup>a</sup> Size of original form, 4 inches wide by 7 inches long, on heavy flexible stock.

<sup>b</sup> Original form is 12½ inches high by 14½ inches long. Space for five names instead of one is provided, and columns for each day of the month. Each sheet is kept in time office until the end of the month, when it is turned in to the main office for balancing and checking.

<sup>c</sup> Size of original form, 3 by 5 inches.



[On reverse side.]

	Amount.	Signed.
Store		
Board		
Rent		

NOTICE.—The above must be filled in with ink; if there is no amount write "None" and sign name.

FORM 19.—*Slip handed by timekeeper to employee who leaves during month.<sup>a</sup>*

SANTA RITA MINE,

191....

Name..... P. R. No. ....

	Amount.
..... shifts, @ \$.....	\$.....
..... shifts, @ \$.....	.....
..... shifts, @ \$.....	.....
..... shifts, @ \$.....	.....
Total amount earned.....	.....
DEDUCTIONS.	
Doctor and hospital.....	\$.....
Insurance.....	.....
Store.....	.....
Board.....	.....
Rent.....	.....
.....	.....
.....	.....
Total deductions.....	.....
Balance due.....	.....

The above statement is correct.

Signed.....

FORM 20.—*Personal record card.<sup>b</sup>*

Name Date employed No.  
 Address Boards  
 Dependents  
 Age Weight Height Eyes Hair M. or S.  
 Nationality Engaged by  
 Last employer  
 Education

Date  
 Dept.  
 Occ.  
 Rate

Date  
 Dept.  
 Occ.  
 Rate

Left employ—Date

Reason:

<sup>a</sup> Size of original form, 5½ by 6 inches.<sup>b</sup> Original card, ruled cardboard, 6 inches wide by 4 inches high.

## ACCOUNTING.

When the labor-distribution cards (Form 15) have been signed and punched by the foreman and handed in to the main office they are entered on loose sheets similar to Form 21, which are kept in binders, forming large loose-leaf books. The labor data are all grouped, so that one or more of these sheets is used for firemen of all kinds, one or more for engineers, one or more for drillers, and so on. Each fireman, engineer, driller, etc., is charged against the particular locomotive or steam shovel fired by him for each day of his service during the month. These forms thus filled out present a complete daily record of labor charges classified and distributed against certain accounts. When this daily record has been summed up at the end of the month, the totals from it are entered on the Santa Rita labor-distribution sheets, the character of which is indicated in Form 22. This shows the total labor charges for the month properly distributed and arranged. One or more of these sheets is used for accounts that may be grouped under the general term, "Operation." In this group the columns are headed: Steam shovel 1, etc.; locomotives 1, etc.; drill 1, etc., and dump cars. The next group of headings on these sheets cover the following accounts: Rents and repairs, light service, automobile expense, prospect-drill expense, hospital fund, special water investigation. These are general ledger accounts and are carried in the general office at Hurley.

FORM 21.—Form used in recording data on labor-distribution cards.<sup>a</sup>

Month.....191..

Account.....

Date.	Shifts.	Amount.	Shifts.	Amount.	Total amount.
1					
2					
3					
4					
5					
6					
29					
30					
31					
Total					

<sup>a</sup> Original form 11 inches high by 26 inches wide, with wide margin on left for binding; 20 instead of 2 of the "Shifts-Amount" columns appear on the original, and space is provided for 31 days of the month.

FORM 22.—Form used in recording total labor charges for each month, showing distribution.<sup>a</sup>

## SANTA RITA LABOR DISTRIBUTION.

Sheet No. ....

.....191.

Occupation.					
1 Assayers and samplers					1
2 Boilermakers and helpers					1
3 Brakemen					1
4 Blacksmiths and helpers					1
5 Blasters					1
6 Compressormen					1
7 Chauffeur					1
8 Car repairers					1
9 Central					1
10 Carpenters and helpers					1
11 Cranemen					1
12 Drill foreman					1
13 Drillers					1
14 Drill helpers					1
15 Drill laborers					1
16 Drill mechanics					1
17 Dumpmen					1
18 Engineers and surveyors					1
19 Electricians					1
20 Engineers, locomotive					2
21 Engineers, steam shovel					2
22 Engineers, stationary					2
23 Firemen, locomotive					2
24 Firemen, steam shovel					2
25 Firemen, stationary					2
26 Foremen, steam shovel					2
27 Foremen, tracks					2
28 Foremen, surface					2
29 Laborers and bosses					3
30 Masons and plasterers					3
31 Master mechanics					3
32 Machinists and helpers					3
33 Machinemen and helpers					3
34 Miners					3
35 Oilman and helpers					3
36 Officemen					3
37 Pumpmen					3
38 Plumbers and pipe fitters					3
39 Painters					3
40 Pitmen					4
41 Repairmen					4
42 Samplers					4
43 Switchtenders					4
44 Stableman					4
45 Sandmen					4
46 Teamsters					4
47 Topmen					4
48 Trammers					4
49 Timber framers					4
50 Trackmen					5
51 Watchmen					5
52 Water and coal tenders					5
53 Yardmasters					5
54					5
55					5
Total					
1					
2					
3					
4					
5					
Total					
Ore					
Waste					

<sup>a</sup> Original form 20½ inches wide by 34 inches high. Original form has 20 instead of 4 distribution columns.



Another group of headings on these sheets indicate the following accounts: Water supply, Santa Rita boiler, compressor expense, heating, machine shop, blacksmith shop, carpenter shop, light service, engineering and surveying, assaying and sampling, office expense, pit drainage, stable expense, superintendent and general labor, and a few other accounts. Another group of accounts comes under the general term "Additions to plant," and the individual columns are headed "Crusher bins," "Hospital addition," etc. A fifth group is known as surface-development accounts. They include surface drainage, additions to dump lines, company roads, etc. The accounts-recoverable group comprises mess accounts, county roads built or repaired, etc.

Each column on each of these sheets (Form 22) represents a separate account. When these columns have been added, the total for each is divided between ore and waste according to the amount of time spent by each shovel in digging ore and the time spent in digging waste. These time figures are obtained from the reports of the shovel engineers, and are checked against foremen's reports, etc.

From the data recorded on the other forms the operating-cost sheets for the month (Form 23) are made out. One of these is used to show mining costs and another to show stripping costs. The expense apportioned to each is obtained from the total time spent by the shovels in digging waste and the total time spent by them in digging ore. The percentage of the total time spent in digging ore multiplied by the total cost for the month gives the cost of mining for the month. The remainder is the total cost of stripping. The recapitulation at the bottom gives a complete summary of the cost of production to date as well as costs for drilling, blasting, loading, and hauling. Thus, Form 23 provides a convenient and concise summary of all the costs.

Form 23.—Operating-cost sheet. a

	Steam shovel No. 1.	Steam shovel No. 10.				Locomotive No. 1.	Locomotive No. 20.	Dump cars.	Air drilling.	Churn drilling.		Total cost.	Average per.....
LABOR.													
Bankmen													
Boilermakers													
Blacksmiths													
Blasters and powdermen													
Brakemen													
Carpenters													
Car trimmers													
Car repairs													
Caulkers													
Drillers and helpers													
Drummen													
Electricians													
Engineering													
Engineers													
Firemen													
Floating gang													
Foremen													
Machinemen and helpers													
Machinists													
Office men													
Oilmen													
Pipelitters													
Pitmen													
Pumpmen													
Sandmen													
Switchtenders													
Teamsters													
Track gang													
Watchmen													
Water and coal tenders													
Yardmaster													
Assaying and sampling													
Stable expense													
Superintendent and general labor													

Compressor expense	
Water supply	
Surface expense	
S. R. pumping and repairs	
Total labor	
Cost per	
SUPPLIES.	
Caps, fuse, and explosives	
Drill steel	
Electrical supplies	
Fuel	
Gasoline	
Miscellaneous	
Oil, waste, and packing	
Pipe and fittings	
Repairs and renewals	
Tools	
Teaming	
Office expense	
Engineering expense	
Assaying and sampling	
Stable expense	
Superintendent and general labor	
Compressor expense	
Water supply	
Surface expense	
S. R. pumping and repairs	
Total supplies	
Cost per	
Total labor and supplies	
Cost per	
No. cars loaded	
No. .... loaded	

<sup>a</sup> Original sheet, 17 by 31½ inches.





## WAREHOUSE METHODS.

The importance of the warehouse or storehouse division will be grasped from the fact that it always carries on hand and accounts for about a quarter of a million dollars' worth of stock of most varied character, say from turpentine to teeth for steam shovels.

All invoices received during the month are recorded in triplicate, Form 24, a "statement of invoices." The original of these is sent to the cashier's office, a duplicate goes to the supply agent, and the triplicate is retained in the warehouse. The invoices are then entered on the "warehouse-record" sheets (Form 25). These give a record of the freight charges, the material actually received, and the material invoiced. From this form the material is entered on large "commodity distribution" sheets, where the data covering different commodities are segregated in different columns, so that the amount of each received is easily determined. The commodities are then entered in the "price-and-invoice" book (Form 26), each page being given over to a different article, such as cylinder oil or steel rails. The price and balance on hand is kept posted for each month, and in addition stock is taken in December of each year and checked against this book.

Data covering the supplies issued are carefully recorded in several ways. Form 27 is an order on the storekeeper for material. The data on each of these forms received are entered in the "supplies issued record" book (Form 28), which shows the kind, quantity, and cost of supplies issued for every day of each month. At the end of the month these daily records are summed up on a monthly record known as the "Santa Rita supply distribution" sheet (Form 29). This summarizes the supplies charged during the month and furnishes part of the data for the work of the cost accountant in the main office.

The warehouse also keeps a record of the powder supply. Every week the powder supply is balanced in pounds of weight and every month in money. Form 30 is used for this record.

Many other labor-saving forms are used, but as they pertain more to general warehouse accounting than to mine accounting, they are not introduced here.







FORM 29.—Form used in summarizing daily records from Form 28.<sup>a</sup>

## SANTA RITA SUPPLY DISTRIBUTION.

Sheet No. ....

191..

	On hand. (b)	Received. (c)	Used. (c)	On hand. (d)	
1 Air-drill repairs					1
2 Air-brake repairs					2
3 Babbitt and metals					3
4 Belting					4
5 Boiler flues					5
6 Boiler compound					6
7 Blacksmith coal					7
8 Burlap and canvas					8
9 Candles					9
10 Cement, brick, and lime					10
11 Castings					11
12 Caps					12
13 Coke					13
14 Drill repairs					14
15 Dump-car repairs					15
16 Electrical fixtures and supplies					16
17 Fuse					17
18 Fire brick and clay					18
19 Fuel					19
20 Furniture and fixtures					20
21 Grease and lubricants					21
22 Hay, grain, and feed					22
23 Hose and fittings					23
24 Hardware					24
25 Iron and steel					25
26 Illuminating oils					26
27 Laboratory supplies					27
28 Lumber					28
29 Locomotive repairs					29
30 Labor					30
31 Machinery repairs					31
32 Metal lath					32
33 Nuts, bolts, and washers					33
34 Nails					34
35 Packing					35
36 Pipe and fittings					36
37 Pulleys and shafting					37
38 Powder					38
39 Paint, tar, and glass					39
40 Pump repairs					40
41 Rail and fittings					41
42 Rope and cable					42
43 Screens					43
44 Steam-shovel repairs					44
45 Stationery					45
46 Structural steel					46
47 Sand					47
48 Switching					48
49 Ties					49
50 Teaming					50
51 Timber and lagging					51
52 Tools					52
53 Waste					53
54					54
55					55
Total					
Santa Rita store supplies					
Electric power from Hurley					
1					1
2					2
3					3
4					4
5					5
Total					
Ore					
Waste					

<sup>a</sup> Original form, 24 by 24 inches. Original has 20 instead of 2 blank columns for headings showing distribution of supplies.<sup>b</sup> At first of month.<sup>c</sup> During month.<sup>d</sup> At first of succeeding month.





## **PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF METAL MINING.**

Limited editions of the following Bureau of Mines publications are temporarily available for free distribution. Requests for all publications can not be granted, and applicants should select only those publications that are of special interest to them. All requests for publications should be addressed to the Director, Bureau of Mines, Washington, D. C.

BULLETIN 53. Mining and treatment of feldspar and kaolin in the southern Appalachian region, by A. S. Watts. 1913. 170 pp., 16 pls., 12 figs.

BULLETIN 62. National mine-rescue and first-aid conference, Pittsburgh, Pa., September 23-26, 1912, by H. M. Wilson. 1913. 74 pp.

BULLETIN 75. Rules and regulations for metal mines, by W. R. Ingalls, James Douglas, J. R. Finlay, J. Parke Channing, and John Hays Hammond. 1915. 296 pp., 1 fig.

BULLETIN 80. A primer on explosives for metal miners and quarrymen, by C. E. Munroe and Clarence Hall. 1915. 125 pp., 15 pls., 17 figs.

BULLETIN 101. Abstracts of current decisions on mines and mining, October, 1914, to April, 1915, by J. W. Thompson. 1915. 138 pp.

TECHNICAL PAPER 4. The electrical section of the Bureau of Mines, its purpose and equipment, by H. H. Clark. 1911. 12 pp.

TECHNICAL PAPER 6. The rate of burning of fuse as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigations of fuse and miners' squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

TECHNICAL PAPER 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp., 1 fig.

TECHNICAL PAPER 15. An electrolytic method of preventing corrosion of iron and steel, by J. K. Clement and L. V. Walker. 1913. 19 pp., 10 figs.

TECHNICAL PAPER 17. The effect of stemming on the efficiency of explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp., 11 figs.

TECHNICAL PAPER 18. Magazines and thaw houses for explosives, by Clarence Hall and S. P. Howell. 1912. 34 pp., 1 pl., 5 figs.

TECHNICAL PAPER 19. The factor of safety in mine electrical installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

TECHNICAL PAPER 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

TECHNICAL PAPER 30. Mine-accident prevention at Lake Superior iron mines, by D. E. Woodbridge. 1913. 38 pp., 9 figs.

TECHNICAL PAPER 40. Metal-mine accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 54 pp.

TECHNICAL PAPER 41. Mining and treatment of lead and zinc ores in the Joplin district, Missouri; a preliminary report, by C. A. Wright. 1913. 43 pp., 5 figs.

TECHNICAL PAPER 46. Quarry accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 32 pp.

TECHNICAL PAPER 47. Portable electric mine lamps, by H. H. Clark. 1913. 13 pp.

TECHNICAL PAPER 58. The action of acid mine water on the insulation of electric conductors; a preliminary report, by H. H. Clark and L. C. Ilsley. 1913. 26 pp., 1 fig.

TECHNICAL PAPER 59. Fires in Lake Superior iron mines, by Edwin Higgins. 1913. 34 pp., 2 pls.

TECHNICAL PAPER 61. Metal-mine accidents in the United States during the calendar year 1912, compiled by A. H. Fay. 1913. 76 pp., 1 fig.

TECHNICAL PAPER 62. Relative effects of carbon monoxide on small animals, by G. A. Burrell, F. M. Seibert, and I. W. Robertson. 1914. 23 pp.

TECHNICAL PAPER 67. Mine signboards, by Edwin Higgins and Edward Steidle. 1913. 15 pp., 1 pl., 4 figs.

TECHNICAL PAPER 77. Report of the Committee on Resuscitation from Mine Gases, by W. B. Cannon, G. W. Crile, Joseph Erlanger, Yandell Henderson, and S. J. Meltzer. 1914. 36 pp., 4 figs.

TECHNICAL PAPER 92. Quarry accidents in the United States during the calendar year 1913, compiled by A. H. Fay. 1914. 76 pp.

TECHNICAL PAPER 94. Metal-mine accidents in the United States during the calendar year 1913, compiled by A. H. Fay. 1914. 73 pp.

TECHNICAL PAPER 100. Permissible explosives tested prior to March 1, 1915, by S. P. Howell. 1915. 15 pp.

MINERS' CIRCULAR 5. Electrical accidents in mines, their causes and prevention, by H. H. Clark, W. D. Roberts, L. C. Ilsley, and H. F. Randolph. 1911. 10 pp., 3 pls.

MINERS' CIRCULAR 8. First-aid instructions for miners, by M. W. Glasgow, W. A. Raudenbush, and C. O. Roberts. 1913. 67 pp., 51 figs.

MINERS' CIRCULAR 10. Mine fires and how to fight them, by J. W. Paul. 1912. 14 pp.

MINERS' CIRCULAR 11. Accidents from mine cars and locomotives, by L. M. Jones. 1913. 16 pp.

MINERS' CIRCULAR 13. Safety in tunneling, by D. W. Brunton and J. A. Davis. 1913. 19 pp.

MINERS' CIRCULAR 15. Rules for mine-rescue and first-aid field contests, by J. W. Paul. 1913. 12 pp.

MINERS' CIRCULAR 17. Accidents from falls of rock or ore, by Edwin Higgins. 1914. 15 pp., 8 figs.



# INDEX.

A.		Page.			Page.
Accidents from blasting.....		58	Car samples of ore, taking of.....		36
mine, number of.....		97	Cars, costs of operating, form for recording..	108-110	
reporting of, form for.....	95, 96		cost of repairing.....		65
Accounting methods, forms used in.....	106, 108-110		description of.....		64, 65
"Adobe" blasting. <i>See</i> "Dobe" blasting.			view of.....		64
Air, operation of drills by.....	42		Carasco, José Manuel, mining of Santa Rita		
on cost of.....	45		ore by.....		7
form of recording.....	108-110		Cartridges, precautions in handling.....		58
Air-compressor plant. <i>See</i> Compressor plant.			Caving in churn-drill sampling, remedy for...		11
Assay plan of ore body, making of.....	19		Chalcocite in Santa Rita district, occurrence		
Assay shop, equipment for.....	37		of.....		4
forms used in.....	38-40		Chalcopyrite in Santa Rita district, occur-		
methods.....	35-38		rence of.....		4
work done at.....	85		Charging of drill holes, equipment for.....		54
Assaying, cost of.....	37, 38		triple, danger from.....		53
Assays, gold, making of.....	36		Churn-drilling at Santa Rita.....		5, 6
<i>See also</i> Drill-hole samples; Drilling.			accuracy of, tests of.....		11
Aztecs, mining of copper ore by.....	7		<i>See also</i> Prospect drilling.		
B.			Churn drills, costs of operating, form for re-		
Bank holes, charges for.....	52		cording.....	108-110	
exploding, view of.....	48		details of.....		42
figure showing.....	53		efficiency of.....		13, 14
definition of.....	51		sampling of ore by.....		42
results of firing, figure showing.....	55		errors in.....		10, 11
"springing" charges in, figure showing.....	51		types of.....		10, 42
Benchmarks, distance between.....	41		water used by.....		75
estimation of ore by.....	42		<i>See also</i> Cyclone drills.		
height of, factors governing.....	41		Clark cars for transporting ore, views of.....		64
view of.....	42		<i>See also</i> Cars.		
Bingham, Utah, mining of copper ores at.....	1		Coal used in steam shovels, amount of.....		63
Black powder, handling and storing of.....	58, 59		Colorimetric determinations of samples.....		36, 38
quantity used in blasting.....	56		Compressor plant, equipment for.....		70
Blast-hole samples, taking of.....	36		Copper in ore, critical content of.....	16, 19, 20	
Blast holes, drilling of.....	43-45		determination of.....		32
cost of.....	45		<i>See also</i> Metallic copper; Ores, copper.		
extent of.....	42		Copper ores in Santa Rita district.....		4
<i>See also</i> Drill holes.			<i>See also</i> Ore bodies.		
Blasting, accidents from.....	58		Costs, blasting.....		57
average costs of.....	78		mining.....		28, 30
in metal mines, purpose of.....	46		factors governing.....		19
time for, factors governing.....	53		operating form for recording.....	108-110	
Blasting machines, testing of.....	58		prospect drilling.....	14, 44, 45	
Blasting powder. <i>See</i> Black powder; Trojan			smelting.....		33
powder.			<i>See also</i> Blasting; Cars; Crushing plant;		
"Block holing," drill used for.....	45		Fuel; Hauling; Labor; Loading; Loco-		
Boiler scale, composition of.....	74		motives; Milling; Mining; Power;		
removal of.....	73		Power plant; Steam shovels; Strip-		
"Bulldozing." <i>See</i> "Dobe" blasting.			ping; Waste.		
C.			Cotton waste used on steam shovels, amount		
C. Y. B. ore body, map showing.....	6		of.....		63
ore mined in.....	27		Crushing plant, cost of.....		68
Caps, blasting, handling and storing of.....	58, 59		details of.....		67
			location of.....		6, 67
			transportation to.....		65, 66
			view of.....		68



	Page.		Page.
Cuprite in Santa Rita district, occurrence of.....	4	Estrella ore body, map showing.....	6
Cyclone drills, efficiency of.....	43, 44	ore mined in.....	27
merits of.....	10	Excavation, cross section of, figure showing..	88
operation of.....	43, 44	estimates of, form for.....	87
speed of.....	42	extent of, computation of.....	89
D.		plan map of, figure showing.....	88
Detonators, blasting in metal mines by.....	47	records of, reports of.....	89
exploding charge by, figure showing.....	54	view of.....	70
precautions in handling.....	58, 59	<i>See also</i> Steam shovels.	
<i>See also</i> Electric detonators.		Explosives, costs of.....	57
Diamond drilling at Santa Rita mines.....	8	for metal mines.....	58
Diamond drills, relative merits of.....	10	efficiency of, factors determining.....	55
"Dobe" blasting, costs of.....	57	kinds of.....	46
definition of.....	57	handling of.....	48, 58, 59
economy of.....	46	quantity of, relation to rock broken.....	56
objections to.....	57	thawing of.....	48
placing of charges for.....	57	Explosives magazine, details of.....	47
view of.....	54	entrance to, view of.....	48
precautions in.....	58	figure showing.....	47
Drainage in Santa Rita district.....	6	F.	
equipment for.....	68	First-aid work, equipment for.....	97, 98
factors affecting.....	68, 69	Foster, L. E., acknowledgments to.....	2
Drill-hole records, calculation of ore from.....	20-23	formulas for estimating ore by.....	28
Drill-hole samples, assay record of.....	14, 15	Fuel used at Santa Rita, cost of.....	76
evaporation of, furnace for, figure showing	35	storage of.....	76
method of preparing.....	35, 36	<i>See also</i> Coal.	
Drill holes, chambering of, purpose of.....	48	Fuse for explosives, types of.....	47
charge for, factors determining.....	51	for primer, arrangement of, figure showing.	50
charging of.....	51-53	G.	
firing of.....	54, 55	Gold in copper ore, assays of.....	36
results of firing, figure showing.....	56	Gopher blasting, description of.....	60-62
sounding of.....	49	explosives used in.....	60, 62
"springing," charges for.....	49	placing of holes for, figure showing.....	61
exploding of, view showing.....	48	preparation for.....	60
procedure in.....	50	purpose of.....	59
purpose of.....	48	tamping of.....	62
types of.....	51	yardage removed by.....	62
<i>See also</i> Prospect holes.		H.	
Driller, reports of, form for.....	12, 13	Hammer drills, mining of ore by.....	42
Drilling of ore, methods of.....	10	operation of.....	45, 46
reports of, distribution of, figure showing.	11	Haulage, average costs of.....	78
samples from, assaying of.....	11	Hayes, M. B., mining at Santa Rita by.....	8
<i>See also</i> Churn drilling; Diamond drill-		Hearst ore body, map showing.....	6
ing; Prospect drilling.		ore mined in.....	27
Drills. <i>See</i> Churn drills; Hammer drills;		Hospital, cases treated by.....	95
Tripod drills.		fees for.....	94
Dumps, method of sampling.....	40	maintenance of.....	94
Dynamite for blasting, use of.....	46	Hurley, N. Mex., assay shop at.....	85
E.		mill at, accidents in.....	97
Electric detonators for use in metal mines...	47	capacity of.....	83
precautions in handling and storing.....	59	costs at.....	85, 87
testing of.....	59	details of.....	79, 81
Electric wires for firing charge, arrangement		flow sheet of, figure showing.....	79, 81
of, figure showing.....	54	operation of.....	78, 82, 83
Electricity for springing holes.....	50, 51	power required for.....	83
Elguea, Manuel Francisco, mining at Santa		transportation to.....	78
Rita by.....	7	pumping stations at, equipment of.....	84
Employees, discharge of, form used in.....	101, 103	repair shops at, equipment of.....	86
personal record of, form used for.....	104	water supply at.....	84
special, "time" of.....	101	I.	
"time" of, forms for recording.....	100, 102, 103	Ingersoll-Rand drill, operation of.....	45
Engineering department, compilations by...	91		
forms used by.....	87, 89-94		
work of.....	87		

	J.	Page.			Page.
Janney, W. H., acknowledgments to.....	2		Ores, copper, classification of.....		15
Jointing shown by excavation, view of.....	70		coarse, crushing of.....		66-68
			formula for determining.....		28
			loading of. <i>See</i> Steam shovels.		
			method of treatment, formulas for		
			determining.....		31-33
			mined by steam shovels, proportion of, to		
			waste.....		27
			smelting, formula for determining ..		30, 31
			profits from.....		34
			transportation of. <i>See</i> Cars; Loco-		
			motives.		
			copper in, critical content of.....		16, 19, 31, 32
			methods of estimating.....		19-40
			figure showing.....		17
			Ore bodies at Santa Rita, character of .....		4, 5
			cross section of, compilation of.....		16
			description of.....		16, 17
			form for.....		18
			delimiting of, factors governing.....		18
			depth of, effect on mining methods.....		18
			division into triangles, figure showing ..		23
			method of.....		23
			occurrence of.....		3
			specific gravity of, determination of.....		25
			total ore in, calculation of.....		24, 25
			Ore stock pile, ore transported to.....		65, 66
			form for recording.....		93
			Organization chart, view of.....		42
			Overburden at Santa Rita, character of.....		5, 6
			depth of, as factor in delimiting ore bodies.		18
				P.	
			Paige, Sidney, acknowledgments to.....		2
			Permanganate method, assay of samples by.		38
			Pit samples, form for reporting.....		39
			taking of.....		36
			Powder, supply of, form for recording.....		115
			<i>See also</i> Black powder; Quarry powder;		
			Trojan powder.		
			Power, costs.....		74
			distribution of.....		70-72
			Power plant, details of.....		70, 71
			equipment of.....		70, 71
			operation of, cost of.....		71, 72
			details of.....		71
			"Primer," arrangement of fuse for, figure		
			showing.....		50
			definition of.....		50
			Prospect drilling, cost of.....		14
			<i>See also</i> Churn drilling; Drilling.		
			Prospect hole samples, method of preparing.		35
			<i>See also</i> Drill holes.		
			Prospecting in Santa Rita district, methods		
			of, factors determining.....		5, 6
			<i>See also</i> Churn drilling.		
			Pumps at Hurley mill, operation of.....		84
			Pyrite in copper ore, effect of.....		5
				Q.	
			Quarry powder, use of.....		46
				R.	
			R. W. T. ore body, map showing.....		6
			ore mined in.....		27
			Recreation at Santa Rita, facilities for .....		98

	Page.		Page.
Repair plant, crew for.....	73	Steam shovels, crew of.....	63
equipment for.....	73, 74, 86	"time" of, form for recording.....	100
Rescue work at Santa Rita, equipment for..	97	description of.....	63
training for.....	98	fuel used by.....	63
Rock, grade of, for mining.....	16, 19	mining with.....	6, 18
formula for determining.....	28	operation of, form for recording.....	92
results of blasting in.....	56	results of.....	90
"tightness" of, as affecting blasting efficiency		ore mined by.....	27, 63
figure showing.....	55	water used by.....	75
figure showing.....	56	<i>See also</i> Excavations.	
<i>See also</i> Ore.		Stripped areas, measurement of.....	37
Romero mine, N. Mex., early working of....	8	Stripping, cost of.....	37, 78
		Sullivan drill, operation of.....	45
S.		Sully, J. M., acknowledgments to.....	2
Safety rules for mining at Santa Rita.....	58, 59	sampling by.....	9, 40
Sample shop, equipment for.....	37	Sulphide ores of copper, character of.....	4
Samples, form for reporting.....	40	overburden of.....	5
<i>See also</i> Blast-hole samples; Car samples;		Supplies, cost of.....	14
Drill-hole samples; Pit samples;		form for recording.....	108-110
Prospect-hole samples; Steam-		distribution of, form for recording.....	114
shovel samples.		issuance of, form for.....	113
Sampling of ore, errors in.....	10, 11	records for, forms for.....	111
methods for.....	10, 35-37		
Sanitation at Santa Rita.....	97	T.	
Santa Rita district, N. Mex., accidents at...	97	Thorne, Harry, acknowledgments to.....	2
climate of.....	2, 3	Time-keeping methods, forms used in.....	99-104
drainage in.....	5, 7, 68	Toe holes, charges for.....	52
geologic conditions in.....	3, 4	charging of.....	53, 54
location of.....	2	definition of.....	51, 52
maps showing.....	2	loading of, view of.....	54
mine workings at, description of.....	6, 7	results of firing, figure showing.....	55
extent of.....	9	Tracks for transporting ore, description of...	66
patents issued for.....	8	maintenance of.....	66
mining in.....	1, 7-9	Transit book, records in, form for.....	87
costs of.....	30, 31	Triangular-prism method of estimating ore...	20-27
ore bodies in. <i>See</i> Ore bodies.		calculations in.....	20-22, 24-26
prospecting at.....	3	Tripod drills, sharpening of.....	45
railroad facilities of.....	2	types of.....	45
Shovel powder houses, details of.....	48	Trojan powder, blasting by means of.....	46
Sierra ore body, map showing.....	6		
ore mined in.....	27	W.	
Siqueiros, —, mining at Santa Rita by.....	8	Warehouse accounts, method of keeping.....	111
Smelter, ore transported to.....	65, 66	form for.....	112
form for recording.....	93, 94	Waste from Santa Rita mines, deposition of.	7
Smelting ore, determination of, formula for...	30, 31	transportation of.....	65, 66
Spencer, A. C., acknowledgments to.....	2	cost of.....	66
"Springing" drill holes. <i>See</i> Drill holes.		form for recording.....	92
"Springing gage," cross sections of, figure		Water-softening plant, flow sheet of, figure	
showing.....	49	showing.....	77
description of.....	49	location of.....	76
method of using, figure showing.....	50	operation of.....	76
Steam-shovel computation book, entries in..	89	Water supply, adequate, need of.....	75
Steam-shovel samples, form for reporting....	39	at Hurley mill.....	74, 75
taking of.....	36	at Santa Rita, equipment for, details of.	75
Steam shovels, costs.....	64	Whitney, J. P., mining at Santa Rita by...	8
form for recording.....	108-110		

DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

MELTING ALUMINUM CHIPS

BY

H. W. GILLETT AND G. M. JAMES





The Bureau of Mines, in carrying out one of the provisions of its organic act—to disseminate information concerning investigations made—prints a limited free edition of each of its publications.

When this edition is exhausted copies may be obtained at cost price only through the Superintendent of Documents, Government Printing Office, Washington, D. C.

The Superintendent of Documents *is not an official of the Bureau of Mines*. His is an entirely separate office and he should be addressed:

SUPERINTENDENT OF DOCUMENTS,  
Government Printing Office,  
Washington, D. C.

The general law under which publications are distributed prohibits the giving of more than one copy of a publication to one person. The price of this publication is 10 cents.

## CONTENTS.

---

	Page.
Introduction.....	5
Nature of scrap aluminum.....	5
Clippings and punchings.....	5
Scrap wire and cable.....	6
Castings, chips, and borings.....	6
Recovery on scrap aluminum.....	7
Methods of melting advocated in the literature.....	11
Adding scrap to molten metal.....	11
Use of fluxes in melting aluminum scrap.....	13
Briquetting borings.....	19
Oxidation of aluminum.....	20
Nitridation of aluminum.....	20
Coalescence of molten aluminum globules.....	22
Analogous problems in coalescence of other metals.....	23
Mercury and tin.....	23
Blue powder.....	24
Summary of methods suggested.....	26
Methods used commercially in melting chips.....	27
Practice at an American plant.....	27
Crucible method.....	27
Melting in iron pots, with puddling and $ZnCl_2$ flux.....	28
Melting with $ZnCl_2$ - $NaCl$ flux in crucibles.....	30
Remelting dross.....	30
Melting with salt and fluorspar in crucible coke furnace.....	31
Experiments in melting aluminum chips.....	32
General method of making tests.....	32
Kind of chips used.....	33
Control of temperature.....	33
Melting with volatile fluxes.....	34
Melting in atmosphere of chlorine-containing gases.....	35
Melting with continuous puddling.....	35
Retorting.....	36
Melting in a reducing atmosphere.....	36
Heating in absence of air.....	37
Using a heel of molten metal.....	38
Conclusions as to results of tests.....	39
Melting with molten cover.....	40
Tests with a commercial flux.....	41
Tests with dehydrated carnallite.....	42
Discussion of results with the two fluxes.....	42
Sprinkling with zinc-chloride solution and dry salt.....	42
Tests with common salt.....	42

## Experiments in melting aluminum chips—Continued.

Melting with molten cover—Continued.	Page.
Tests by a foundry with salt and calcium chloride flux.....	43
Series 1, melting emery grindings.....	43
Series 2, melting dirty borings with sal ammoniac and NaCl-CaCl <sub>2</sub> fluxes.....	44
Series 3, tests on cleaner borings with sal ammoniac and NaCl-CaCl <sub>2</sub> fluxes.....	44
Test with mixture of CaCl <sub>2</sub> and NaCl.....	44
Fluxes containing fluorides.....	45
Tests with various fluoride fluxes.....	45
Tests with salt and fluorspar flux.....	46
Summary of results with various methods of melting.....	48
Experiments with briquetted chips.....	48
Results of melting briquet No. 1.....	49
Results of melting briquet No. 2.....	50
Conclusions as to tests.....	50
Reverberatory melting.....	51
Tests in reverberatory furnace.....	52
Remarks on results of tests.....	54
Electric melting.....	55
Experiments in melting ingot.....	55
Experiments in melting borings and skimmings.....	55
Commercial test of salt and fluorspar flux.....	56
Effect of sizing the borings.....	58
Experiments in washing the chips.....	59
Washing with hot water.....	60
Washing with a water-soluble cutting compound.....	60
Washing with gasoline.....	61
Washing with caustic-soda solution.....	63
Miscellaneous experiments in electric furnaces.....	65
Experiments with indirect arc furnace.....	66
Melting with alundum-magnesite lining.....	66
Experiment with graphite crucible lining.....	69
Experiments with crucible resistor furnace.....	70
Conclusions from results of tests.....	71
Quality of ingot from aluminum chips.....	72
Effect of impurities.....	75
Copper.....	75
Zinc.....	75
Lead.....	77
Tin.....	77
Antimony.....	77
Manganese.....	77
Iron.....	77
Interrelated effects of iron, copper, and zinc.....	78
Silicon.....	78
Composition of ingot from electric furnace.....	79
Necessity for care and cleanliness in collecting chips.....	80
Probable relative usefulness of different melting methods.....	80
Acknowledgments.....	82
Publications on mineral technology.....	83
Index.....	85

# MELTING ALUMINUM CHIPS.

By H. W. GILLETT and G. M. JAMES.

## INTRODUCTION.

In its work on mineral wastes the Bureau of Mines is studying losses in the melting of nonferrous metals and alloys. The greatest of these losses is that of zinc through volatilization in brass melting, but another and considerable loss is that of aluminum and its alloys in the melting of finely divided material.

In order to get some idea of the magnitude of the loss in melting aluminum scrap and to find out what methods of melting were used information was gathered in regard to present practice; also experiments were conducted to obtain some data on the relative merits of different methods. As the experiments could be made on a laboratory scale only, they were not expected to solve the problem entirely, the object in making them being rather—by indicating which methods were of greater promise—to narrow the field over which large-scale tests and comparisons need be made by those commercially interested in the problem of melting scrap aluminum.

## NATURE OF SCRAP ALUMINUM.

Scrap aluminum may be produced in the fabrication of sheet aluminum into various objects, such as cooking utensils, camera frames, and various ornamental goods, yielding sheet clippings or punchings, in the machining of aluminum castings for automobiles, motor boats, cooking utensils, vacuum cleaners, meter cases, and the multitudinous other articles in which cast aluminum is used, yielding borings or chips, or in the foundry, in the chipping and grinding of rough castings, and in the recovery of buttons of metal from foundry sweepings, or from the dross skimmed from the melting pots.

## CLIPPINGS AND PUNCHINGS.

Sheet clippings and punchings do not, as a rule, show a very great melting loss, for they are normally kept clean and are usually not excessively small nor thin, and may without the use of very great pressure be “cabbaged” or pressed into brick-like bundles, which



are considered almost as valuable for foundry use as ingot metal, particularly as it is necessary, in order to obtain good rolling qualities, to use purer metal for sheet than is sold for casting purposes. "Hard-sheet" aluminum, containing upwards of 1 per cent of manganese, must of course be used with due consideration of the content of the alloyed metal.

#### SCRAP WIRE AND CABLE.

Scrap wire and cable is a still purer aluminum than unalloyed sheet aluminum, as the very purest metal attainable is required for high electrical conductivity. Bare wire and cable is a desirable form of scrap, but insulated wire would be hard to handle in remelting. On account of the growing use of aluminum wire of small diameter, that is electrolytically insulated with a film of aluminum oxide,<sup>a</sup> this wire will at some future time form a variety of scrap that will be more or less difficult to remelt.

#### CASTINGS, CHIPS, AND BORINGS.

The great bulk of the scrap aluminum met with at present consists of old castings and of chips and borings obtained in machining castings. Melting down old castings presents no difficulty, as the loss is scarcely greater than with ingot metal. The chips and borings, however, give much trouble, and form the main problem in regard to metal losses in remelting scrap aluminum.

As at least 95 per cent of all cast-aluminum alloys is used for automobile engine beds or crank cases, gear cases, transmission cases, footboards, bodies, steering-wheel spiders, intake and outlet manifolds, hub caps, and miscellaneous trimmings and fittings, chips from a motor-car manufacturing plant may be regarded as typical.

The drill presses and milling machines used on aluminum castings run either with a light cutting oil, kerosene being often used, or with cutting lubricants of the general type of emulsions of oil in a soap solution. The emulsions seem to be coming into increasing use.

If the chips stand in the air while wet with such aqueous solutions, that is, if the solution is not drained off at once, they become superficially oxidized and caked together.

The chips become covered with whatever lubricant is used, soaking it up like a sponge. If they fall on the floor and are swept up without due care to keep them separate from the floor dust, the dust sticks to the oily or moist borings.

If the machining of iron, steel, brass, or white metal is done near by, chips of those metals are likely to become mixed with the aluminum chips. It is the exception and not the rule to find aluminum chips

<sup>a</sup> Skinner, C. E., and Chubb, L. W., The electrolytic insulation of aluminum wire: Trans. Am. Electrochem. Soc., vol. 26, 1914, p. 137.

that are not contaminated with a few per cent of iron chips. The iron necessitates passing the chips over a magnetic separator. On account of the more curly form of chips from aluminum than from most commercial brasses and bronzes, it is harder to effect a complete separation of the iron, particularly from damp or oily chips.

The chips may go to the aluminum foundry of the automobile plant, if it makes its own aluminum castings, or may be sent to various refiners, who either buy the chips outright or refine them into ingots, make a smelting charge, and return the ingots obtained, or the chips may be sold in small lots to some small dealer in scrap metal, who adds them to various other chips until he has enough to melt down himself or sell to a larger operator.

Some of the larger operators state that it is apparently difficult for some of the small junk dealers to resist the temptation to add road dust, foundry sweepings, or other fine dirt to the borings, as 10 per cent or more will readily stick to oily borings and be scarcely detectable even on close inspection.

The size and thickness of the chips depends on the depth of the roughing and finishing cuts. The thickness commonly runs between 0.005 and 0.02 inch.

#### RECOVERY ON SCRAP ALUMINUM.

Foundry wastes, such as spillings recovered in floor sweepings, small fins that work their way into core prints, chippings from the trimming room, and dust from the emery grinders, are all badly contaminated with dirt. The larger pieces are picked out by hand and the rest concentrated by tumbling dry in a stream of air, or in a wet tumbling barrel. The scum of oxide, dirt, and metal that is skimmed from the melting pots is generally remelted in the foundry, usually with the use of sal ammoniac or zinc chloride, and the dross from this remelting is thrown away, sold to a refiner, or dropped into water as soon as skimmed off, and then subjected to wet tumbling.

In melting fine material, such as borings or chips, fine shot from foundry wastes, and dust from grinding wheels, all heavy scrap being excluded, the recovery as ingot metal runs from 40 to 90 per cent of the metal content, the proportion recovered varying with the size and cleanness of the material and with the facilities, experience, and method of the melter.

It is very doubtful if any but the largest and most experienced refiners average more than 65 per cent recovery from the ordinary dirt-laden borings of commerce. One firm organized solely to refine borings to ingot suspended operations after a few months, and another firm has run its aluminum-refining department only intermittently, finding that when aluminum was scarce it was better to stop refining borings because their price rose above that at which a profit could be

made with the percentage of recovery obtained. At such times the borings go to small refiners, or to foundries that refine the borings separately or add them to the regular melts. In many cases no accurate records of costs are kept by such plants, and the operators may be refining at a considerable loss without knowing it.

A prominent aluminum founder states that with pure aluminum selling at 25 cents per pound and copper at 15 cents, the most common casting alloy, "No. 12," containing 8 per cent copper, would sell in new ingot for about  $24\frac{1}{2}$  cents, whereas good No. 12 borings would command about  $12\frac{1}{2}$  cents and the ingot made from the borings would sell at about 22 cents. If the cost of refining, including the cost of fuel, labor, overhead charges, etc., is arbitrarily assumed to be 1 cent per pound of ingot produced, it will be seen that with a 50 per cent recovery on the gross weight of borings bought, the cost of producing a pound of ingot would be 25 cents for metal and 1 cent for refining charges, or 26 cents, which, with metal selling at 22 cents, would result in a loss of 4 cents per pound of ingot, whereas with a 70 per cent recovery the cost would be about 20 cents, netting a profit of 2 cents per pound. With a 60 per cent recovery there would be a slight loss, whereas with an 80 per cent recovery the profit would be over 5 cents per pound. It is assumed that the same price,  $12\frac{1}{2}$  cents, is paid for borings from which only 50 per cent of metal is recovered as for those yielding 80 per cent.

If it be assumed that the average recovery from chips and foundry waste is 65 to 70 per cent, on the metallic basis, and if, as is shown later, borings kept perfectly clean and properly melted will give 90 to 95 per cent recovery, then 20 to 30 per cent of the aluminum that is machined off castings is unnecessarily lost.

The probable amount of aluminum castings used in motor-car construction from July 1, 1914, to July 1, 1915, has been estimated by the general manager of a firm of aluminum founders, whose production alone is over half that amount, to be between 20,000,000 and 25,000,000 pounds. Addition of heavy export orders to the increasing domestic demand will undoubtedly cause the amount used from July 1, 1915, to July 1, 1916, to exceed 25,000,000 pounds.

The percentage machined off from a rough aluminum casting in the finishing process seems, from data obtained by the writer, to average about 15 per cent, a figure which is much higher than was estimated offhand by several people connected with the automobile and aluminum casting industries, whose guesses averaged 3 to 4 per cent. The writer's result was reached by weighing rough and finished castings at an automobile plant making a car in the \$700 to \$800 class, and by weighing rough and finished castings made for various other motor-car manufacturers at an aluminum foundry, with the following results:



*Proportion of metal machined off in finishing aluminium castings for motor cars.*

Casting.	Weight of rough casting.	Weight of finished casting.	Percentage of metal machined off.
	<i>Pounds.</i>	<i>Pounds.</i>	
Gear and transmission case.....	28.50	24.50	14
Small double-flanged exhaust elbows.....	.42	.35	15
Intake pipe for 6-cylinder motor.....	5.00	4.80	4
Gear case for small motor.....	8.90	7.00	21
Gear case for very small motor.....	1.58	1.50	5
Crank case for 8-cylinder motor.....	94.00	80.00	14.5

As crank cases and transmission cases make up the bulk of the weight of aluminum castings in a car, the values obtained from them will have more effect on the total than those obtained from the smaller castings.

The results of this tabulation, which indicate that 15 per cent was machined off, were so at variance with the estimates cited that the matter was taken up with the makers of a car in the \$2,000 class, who state that on that car there are used 47 aluminum parts which are machined, the total weight of the rough castings used per car being 166.19 pounds. The loss per car in chips is 25.15 pounds, or 15.1 per cent. The weights were taken on the average of a number of like parts, in some cases as many as 50 being weighed. On the basis of 15 per cent of metal machined off, the yearly production of chips from aluminum castings in the United States will then run between 3,000,000 and 3,750,000 pounds. If 20 to 30 per cent of this amount is unnecessarily lost, then 600,000 to 1,125,000 pounds of aluminum alloys worth 16 to 25 cents per pound, based on the average market quotations for the last few years, is lost, or, as these figures take no account of the loss in running down foundry wastes, it may be estimated that in all metal worth approximately \$200,000 at average prices for aluminum, is lost each year in the United States through the low recovery in melting down aluminum borings and foundry waste. At the abnormal price of aluminum in 1916 this figure would be doubled or trebled.

This estimate is not too high, as the United States Geological Survey states <sup>a</sup> that in 1913 2,198 tons of secondary aluminum and 2,456 tons of secondary aluminum alloys (90 per cent of the latter being "No. 12" alloy, containing 92 per cent aluminum and 8 per cent copper) were recovered, mostly from aluminum clippings and borings, the whole having a value of \$2,199,480. The total consumption of aluminum in the United States in 1913 was 36,190 short tons.<sup>b</sup>

<sup>a</sup> Phalen, W. C., Bauxite and aluminum: Mineral Resources of the United States for 1913, U. S. Geol. Survey, 1914, p. 16.

<sup>b</sup> Phalen, W. C., Loc. cit.



If the pure aluminum, mostly from clippings, which give low loss on melting, be entirely disregarded and the alloys only be considered, and if it be further assumed that only two-thirds of the quantity of alloys came from borings, there would be about 3,250,000 pounds of secondary aluminum alloys recovered. If this figure represents a 70 per cent recovery of the original metal content in the borings, which is as high as is probable, and if a 90 per cent recovery is possible then two-sevenths of 3,250,000 pounds, or over 1,000,000 pounds, was lost in melting. The average value of the secondary pure aluminum and alloys was about  $23\frac{1}{2}$  cents per pound in 1913, according to the figures of the Geological Survey. If the alloy is assumed to be worth 20 cents, the value of the lost metal figures to about \$200,000 for 1913.

The 1914 figures <sup>a</sup> show 2,791 tons of secondary aluminum and 1,731 tons of recovered aluminum contained in alloys, having a total value of \$1,673,140, or an average of  $18\frac{1}{2}$  cents per pound. It is stated that the greater part of the secondary aluminum was recovered from clippings and borings. Pure aluminum borings are not common, and as refineries commonly term borings from aluminum alloys "aluminum borings," their reports may not have made a clear distinction between pure and alloyed metal.

Although figures for 1914 showed a decrease in amount and value of recovered aluminum alloys as compared with 1913, the high average price of aluminum in 1915 will bring the values of the losses for 1915 to about \$300,000, as in the fall of 1915 the price of aluminum rose to about three times the prices quoted in 1914 and in the spring of 1915.

There are, of course, other possible uses for aluminum alloy chips than as a source for the recovery of ingot metal. Their use for the manufacture of aluminum paint at once suggests itself, but it is doubtful if all alloys of aluminum are malleable enough to produce the very thin flat flakes required in aluminum paint. Pure aluminum scrap could doubtless be so used. One automobile firm, during the period of unusually high prices for aluminum in 1915 and 1916, was forced to find a substitute for the aluminum paint previously used on its motors and found one in finely crushed ferrosilicon. This firm also reported that it could, in this period, obtain a higher price for its aluminum chips from a chemical company than it could from the metal refiners. The chips are evidently used in the preparation of anhydrous aluminum chloride, to be used as a condensing agent in the manufacture of dyes.

---

<sup>a</sup> Phalen, W. C., The production of bauxite and aluminum: Mineral Resources of the United States for 1914, U. S. Geol. Survey, 1915, pp. 183-209.

## METHODS OF MELTING ADVOCATED IN THE LITERATURE.

Various methods of melting down chips are suggested in the periodical and patent literature. One of the earliest comments <sup>a</sup> on the subject is as follows:

## ADDING SCRAP TO MOLTEN METAL.

We recently had an inquiry for a method of recovering the aluminum from the skimmings taken from the casting ladle. The company making the inquiry is in the aluminum-casting business, the parts being mostly for automobile work. One essential was that the recovered metal must be of such degree of purity that it could be used over again for casting. As the quantity of skimmings is large, the waste at present going on is considerable.

There is no way of accomplishing this in a commercial manner except by putting the skimmings into the reduction pots, and even that is objectionable with a great many kinds of alloy skimmings, particularly those containing zinc. It is extremely difficult to get finely divided particles of aluminum to agglomerate, even when fused, owing to the difficulty of breaking the skin which covers each particle. The only method by which it was ever done is that recited by De Ville—namely, melting finely divided metal by using a double chloride of sodium and aluminum, which acts as a flux and permits some of the finely divided aluminum to be made into a button. This operation is, however, so difficult and so unusual that the expense of it is considerably more than the value of the metal recovered.

Sperry <sup>b</sup> states "The best way to melt aluminum scrap, such as sheet or chips, is to have a pot of molten aluminum at low red heat and then to add the scrap to it, pushing it down at once under the surface, so that it is not exposed to the air at all. In this manner there is no oxidation of the aluminum and it can be melted with the minimum waste. The scrap will dissolve in the molten aluminum, and this is an ideal condition." He also states <sup>c</sup> that the chips should be added in small quantities at a time to already molten metal with immediate stirring, and the pot fluxed at the end of the melt with a little zinc chloride.

Vickers <sup>d</sup> suggests practically the same method. Still another description <sup>e</sup> of this general method follows:

*Question.*—We have several barrels of aluminum turnings and borings and would like to know the best method of converting this material into pig form. We understand that under certain conditions of melting a considerable loss of aluminum results.

*Answer.*—The melting of finely divided metals entails considerable work, and furthermore, the material must be handled in a manner which will prevent undue oxidation. The best way to accomplish this is to dissolve the finer metal in a bath of molten metal. A large crucible should be used and the molten metal should be obtained by melting ingots or other bulky stock. In the case of aluminum, the bath should be

<sup>a</sup> Editorial, Recovery of aluminum from skimmings: *Metal Ind.*, vol. 5, 1907, p. 100.

<sup>b</sup> Sperry, E. S., Questions and answers: *Brass World*, vol. 9, 1913, p. 296.

<sup>c</sup> Sperry, E. S., Remelting of aluminum chips or borings; *Brass World*, vol. 6, 1910, p. 278; *Jour. Inst. Metals*, vol. 5, 1911, p. 342.

<sup>d</sup> Vickers, C., How to melt aluminum turnings or borings: *Foundry*, vol. 41, 1913, pp. 119, 420.

<sup>e</sup> Editorial, Recovery of aluminum turnings and borings, problems of the brass founder: *Foundry*, vol. 43, May, 1915, p. 195.

raised to a temperature not above red heat. Charge as many borings at a time as the bath will dissolve, but do not leave any of the borings on top of the bath in a solid state. These should be stirred into the molten metal. Inasmuch as the bath increases with each charge of borings, it will dissolve more each time that additions are made. As the borings will cool the bath, the furnace should be closed from time to time, the metal again heated to redness, when more borings can be added, and this process can be continued alternately until the pot is full. At this stage the metal is cast into ingots, but sufficient should be left in the pot to form another bath. If the mush-like dross gathers on the surface of the metal, add a small piece of fused zinc chloride and stir it on the surface. This will liberate the metal from the dross, which will form a cover for the aluminum. The metal should be skimmed before casting into ingots. Incidentally, this dross is of no value.

In doing this work the borings should be passed through a magnetic separator to remove the iron, and under no conditions should the bath of molten metal be heated above redness at any time, as otherwise the aluminum will attack the silica in the crucible and the metal will contain black specks.

A similar method in which turnings are to be charged into a "heel" of molten metal, but a reverberatory furnace used, not a crucible, has been advocated,<sup>a</sup> though no mention is made of stirring or the use of any flux.

Echevarri<sup>b</sup> says, "Scrap can be remelted by submerging in the molten metal, the loss being only 2 to 3 per cent." This statement evidently refers to heavy scrap.

Coulson<sup>c</sup> has recently described a puddling method, commercially used for the recovery of turnings from an aluminum alloy with 5 to 10 per cent of magnesium, as well as considerable experimental work on the problem. Previous to the experimental work, cryolite had been used as a flux, the recovery with clean turnings varying from 60 to 90 per cent on clean material. In the experiments, turnings were melted in air-tight, electrically heated furnaces, in which the air could be displaced by reducing gases, such as hydrogen. Even in such a nonoxidizing atmosphere 10 per cent was lost, and nitrides were still formed, indicating that air was occluded in the borings. Then the turnings were boiled in a 4 per cent salt solution to displace occluded air and wash off dirt. These treated borings, melted under hydrogen, still gave 8 per cent loss. Melting under hydrogen was considered impractical for commercial work, so melting in open crucibles was tried. The borings were washed in benzine and then boiled in a salt solution. The damp borings were fed into a hot crucible containing a small "heel" of molten metal, and each addition thoroughly puddled until the mass became uniformly viscous. After all the borings were in, the charge was allowed to stand until the dross had risen, when the clean metal was poured from beneath the dross, or tapped off from a hole in the bottom of the crucible.

<sup>a</sup> F. H. H., Refining scrap: Foundry Trade Jour., vol. 14, 1914, p. 29.

<sup>b</sup> Echevarri, J. T. W., Aluminum and some of its uses: Jour. Inst. Metals, vol. 1, 1909, p. 128.

<sup>c</sup> Coulson, J., Reclamation of magnalium from turnings: Trans. Am. Inst. Metals, vol. 9, 1915.



Calcium chloride was then stirred into the dross. If the temperature rose through thermit action, powdered cryolite was stirred in. The metal separated from the dross was then poured or tapped. The loss by this method averaged 6 per cent for small melts, and with 200-pound melts of clean material the loss has been as low as 1 per cent.

It is stated that the addition of 1 per cent of calcium or 0.5 per cent of calcium aluminum silicide restores the reclaimed metal to its original physical state as shown by tensile tests.

Such additions should be made with caution, since for some purposes the presence of alkali or alkaline earth metals is detrimental, and in the production of aluminum every effort is made to keep them out, as they are considered highly undesirable impurities. Le Chatelier<sup>a</sup> has shown that calcium aluminum alloys are liable to disintegrate, apparently spontaneously.

Puddling methods were used by De Ville on a small scale in 1859, and his directions, as quoted by Richards,<sup>b</sup> quite closely describe the puddling method in use to-day. These directions are as follows:

When it is desired to melt pieces together, they can be united by agitating the crucible or compressing the mass with a well-cleaned cylindrical bar of iron. Clip-pings, filings, etc., are melted thus: Heat the divided metal to as low a heat as possible—just sufficient to melt it. The oil and organic matter will burn, leaving a cinder, which hinders the reunion of the metal if the mass is not pressed firmly with the iron bar.

#### USE OF FLUXES IN MELTING ALUMINUM SCRAP.

Zinc chloride ( $\text{ZnCl}_2$ ) is often used as a flux in running down chips and skimmings, as well as in ordinary melting, it being claimed that the zinc chloride aids the separation of the metal from the dross so that instead of a pasty mass of metal plus dross only a dry, crumbly dross is taken off.<sup>c</sup> The value or uselessness of zinc chloride as a flux for ordinary melting is a matter of contention among aluminum founders. Lane<sup>d</sup> found that 0.015 per cent of zinc was taken up by an aluminum alloy on one melting and 0.085 per cent after the eighth remelt, zinc-chloride flux being used each time. He suggests that some of the zinc chloride may be decomposed by the heat and part of the zinc may alloy with the aluminum, the nascent chlorine being the active agent of the flux.

Micks<sup>e</sup> states—

The flux that has given the best results in aluminum is chloride of zinc. The zinc combines with the oxygen which is taken up from the aluminum oxide and forms zinc

<sup>a</sup> Le Chatelier, H., *Alterability of aluminum*: *Compt. rend.*, t. 152, 1911, p. 650; *Chem. Abs.*, vol. 5, 1911, p. 1898.

<sup>b</sup> Richards, J. W., *Aluminum, its properties, metallurgy, and alloys*, 1890 pp. 202, 247.

<sup>c</sup> Anon., *Use of zinc chloride in melting aluminum*: *Jour. Inst. Metals*, vol. 2, 1909, p. 320.

<sup>d</sup> Lane, H. M., *Use of magnesium in deoxidizing aluminum alloys*: *Trans. Am. Brass Founders' Assn.*, vol. 4, 1910, p. 101; *Castings*, vol. 7, 1911, p. 164.

<sup>e</sup> Micks, R., *Furnace practice in the brass foundry*: *Brass World*, vol. 11, 1915, p. 150.



oxide, which is then skimmed off together with the aluminum chloride that is also formed in the reaction when the flux is added. The best time to add this flux is after the melting is completed; the surface of the molten metal is then covered with a mass of dross, but when the chloride of zinc is dropped on it in small pieces it will become clear as soon as enough of the flux has been added.

Lane considers that magnesium is a deoxidizer of aluminum. However, Matignon<sup>a</sup> reports the reduction of magnesium oxide by aluminum. It is also stated<sup>b</sup> that although from thermochemical data both magnesium and calcium should reduce aluminum oxide, magnesium does not act as strongly in a thermit mixture as aluminum, and that it probably is not a deoxidizer for aluminum. It was thought that calcium was reduced by aluminum when calcium chloride was used as a flux, and the statement was made that the only value of calcium chloride or zinc chloride as a flux is in disentangling the aluminum from the dross.

Another flux recommended<sup>c</sup> is anhydrous aluminum chloride ( $\text{AlCl}_3$ ), the claim being made that by the use of a small amount of aluminum chloride a recovery of 50 to 70 per cent could be obtained from chips, which without it yielded only 30 per cent.

Another flux recommended consists of sodium carbonate 10 parts, potassium carbonate 2 parts, cryolite 2 parts, and borax 1 part, to be used in the proportion of  $1\frac{1}{2}$  pounds per 100 pounds of chips.<sup>d</sup>

Borax is suggested by Mellen<sup>e</sup> as a flux to dissolve oxides of aluminum.

However, Richards<sup>f</sup> states that aluminum melted under borax is rapidly dissolved, an aluminum borate being formed.

Weber<sup>g</sup> suggests the use of fluorides of zinc, copper, or nickel, mixed with alkali fluorides. The aluminum reduces the fluoride of the heavy metal used, the zinc, copper, or nickel alloying with the aluminum and forming an equivalent amount of aluminum fluoride. The mixture of aluminum fluoride and alkali fluorides then acts as a slag to dissolve the aluminum oxide. Weber also suggests a mixture of zinc chloride and sodium fluoride.

Jones<sup>h</sup> suggests a flux of equal parts of lithium chloride, potassium chloride, and sodium fluoride for use in welding aluminum.

Schoop<sup>i</sup> emphasizes the difficulty of breaking through the thin surface film of oxide in welding aluminum so as to make the fluid metal, beneath the films of the two pieces to be welded, unite, and

<sup>a</sup> Matignon, C., Reduction of magnesia by aluminum: *Compt. rend.*, t. 156, 1913, p. 1157; *Jour. Soc. Chem. Ind.*, vol. 32, 1913, p. 491.

<sup>b</sup> Anon., Oxidation of aluminum: *Foundry*, vol. 37, 1910, p. 225.

<sup>c</sup> Anon., The fusion of aluminum chips: *Jour. Mines Met.*, t. 1, 1912, p. 7; *Chem. Abs.*, vol. 6, 1912, p. 2223; *Brass World*, vol. 9, 1913, p. 365.

<sup>d</sup> Anon., Melting aluminum chips: *Castings*, vol. 38, 1910, p. 78.

<sup>e</sup> Mellen, J. G., and Mellen, W. F., U. S. Patent 982218, Jan. 17, 1911.

<sup>f</sup> Richards, J. W., Aluminum, its properties, metallurgy, and alloys, 1890, p. 81.

<sup>g</sup> Weber, H., Process for melting scrap aluminum or alloys high in the metal: German Patent 242347, Dec. 30, 1910; *Jour. Soc. Chem. Ind.*, vol. 31, 1912, p. 237; *Chem. Abs.*, vol. 6, 1912, p. 2063.

<sup>h</sup> Jones, J. L., Shop problems—fluxing: *Metal Ind.*, vol. 11, 1913, p. 523.

<sup>i</sup> Schoop, M. V., Autogenous welding of aluminum: *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 151.

states that he has tried borax, glass, and other substances to exclude the air, but found that a flux which would dissolve the oxide was needed. He then tried potassium pyrosulphate and many other substances as fluxes to dissolve the oxide, but found a satisfactory solution of the problem only in the use of alkali chlorides. In his patents Schoop<sup>a</sup> mentions a mixture consisting of 60 parts potassium chloride, 20 parts lithium chloride, 12 parts sodium chloride, 8 parts potassium sulphate. In a later patent<sup>b</sup> he substitutes cryolite for the potassium sulphate, and in another,<sup>c</sup> he says, "Experiments have shown that the said defects may be completely avoided by adding fluorides—for example, fluorides of calcium, potassium, or boron to the mixture of chlorides of alkali metals. By means of the improved flux it is possible also to melt down or fuse together in crucibles small and large scraps of aluminum in the shape of castings, sheets, wires, or other objects."

Morrison<sup>d</sup> speaks well of such a flux for welding, as does Seligman.<sup>e</sup> Other welding fluxes have been suggested, Thaulow<sup>f</sup> recommending 96 parts of borax and 4 parts of sodium bisulphate.

Pannell<sup>g</sup> gives the composition of various fluxes for use in welding, as follows:

*Composition of various fluxes for use in welding.*

Flux No.	Constituent.							
	Sodium chloride.	Potassium chloride.	Lithium chloride.	Potassium fluoride.	Sodium fluoride.	Sodium bisulphate.	Potassium bisulphate.	Cryolite.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
1.....	30.0	45.0	15.0	7.0	.....	3.0	.....	.....
2.....	.....	33.0	33.0	.....	33.0	.....	.....	.....
3.....	12.5	62.5	20.8	.....	.....	.....	4.0	.....
4.....	16.0	79.0	.....	.....	.....	.....	5.0	.....
5.....	17.0	83.0	.....	.....	.....	.....	.....	.....
6.....	6.5	56.0	23.5	.....	.....	4.0	.....	10.0

It is seen that these are all mixtures of alkali chloride high in the low-melting potassium and lithium chlorides, with or without the addition of fluorides or bisulphates.

Other processes are based on the theory that as aluminum is readily oxidizable, the melting should take place in the absence of air. Mellen<sup>h</sup> patented a vacuum furnace in which it is claimed that aluminum clippings, turnings, and filings ordinarily melted with a loss of 10 to 40 per cent may be melted with a loss of only 0.5 to 5 per cent.

<sup>a</sup> Schoop, M. V., U. S. Patent 922523, May 25, 1909; English Patent 24096, Oct. 31, 1907; French Patent 374089, Jan. 30, 1907.

<sup>b</sup> Schoop, M. V., U. S. Patent 943164, Dec. 14, 1909.

<sup>c</sup> Schoop, M. V., English Patent 24283, Nov. 2, 1907.

<sup>d</sup> Morrison, W. M., In discussion: Jour. Inst. Met., (British) vol. 1, 1909, p. 150.

<sup>e</sup> Seligman, R., The welding of aluminum; Jour. Inst. Met. (British), vol. 2, 1909, p. 281.

<sup>f</sup> Thaulow, E., U. S. Patent 1139923, Mar. 4, 1914; Brass World, vol. 11, 1915, p. 181.

<sup>g</sup> Pannell, E. V., Recent developments in aluminum; some notes on autogenous welding: Trans. Am. Inst. Metals, vol. 9, 1915. Advance copy, p. 17.

<sup>h</sup> Mellen, G., U. S. Patent 1120732, Dec. 15, 1914.

One inventor, according to parties to whom the invention was offered for \$100,000, had a plan to melt aluminum borings by melting in retorts similar to those used in zinc smelting and passing in steam when it was desired to check the heating and stop oxidation by air. A recovery of 85 per cent was claimed. No further details are available.

Zavelberg <sup>a</sup> suggests for zinc smelting or the recovery of other readily oxidizable metals heating the furnace, discontinuing the heating, charging and closing the furnace, and doing the melting by the stored heat. Practically the same method has been used in the melting of aluminum-magnesium alloys for casting purposes in a tilting open-flame oil furnace, though it does not seem to have been tried on borings.

Richards <sup>b</sup> says:

Fused common salt is used as a flux for aluminum. Fluorspar makes a good flux for the metal, especially in connection with cryolite or common salt, and possesses the property of dissolving the aluminum oxide with which the metal may be contaminated and which, by incrusting small globules, hinders their reunion to a button.

Various fluxes have been used in melting aluminum without having been especially mentioned in connection with borings.

Guertler <sup>c</sup> mentions a mixture of sodium and potassium chlorides.

Zinc chloride, a mixture of sal-ammoniac (ammonium chloride) and common salt, as well as saltpeter (potassium nitrate) are all mentioned.<sup>d</sup>

Schultz <sup>e</sup> also reports favorably on the use of saltpeter in melting aluminum alloys to get sound castings. It seems remarkable that the addition of so strong an oxidizing agent to so readily oxidizable a metal should be beneficial, and the most plausible explanation is that the gas given off by the decomposition of the saltpeter stirs the metal and is carried off with the other and harmful gases previously dissolved in the metal. Saltpeter has not been advocated for use on chips. Sal ammoniac alone is also used.<sup>f</sup>

Gleason <sup>g</sup> melts aluminum under a flux composed of 3 parts fluorspar and 1 part anhydrous boric acid. Mellen <sup>h</sup> uses a flux consisting of salt and sulphur.

Desch <sup>i</sup> mentions borax, which melts at about 650° C., and anhydrous carnallite (equimolecular amounts of potassium and magnesium chlorides), which melts at about 450° C., as forming suit-

<sup>a</sup> Zavelberg, A., U. S. Patent 1136304; German Patents 226257, Feb. 23, 1908; 276364, Aug. 6, 1913. Chem. Abs. vol. 9, 1915, p. 594.

<sup>b</sup> Richards, J. W., Aluminum its properties, metallurgy and alloys, 1890. p. 80.

<sup>c</sup> Guertler, W., Metallographie, Bd. 2, Heft 1, 1913, p. 39.

<sup>d</sup> Editorial, Flux for aluminum alloy—problems of the brass foundry; Foundry, vol. 41, 1913, p. 420.

<sup>e</sup> Schultz, F., In discussion; Trans. Am. Inst. Metals, vol. 8, 1914, p. 132.

<sup>f</sup> Carritto, J. P., Making McAdamite metal; Foundry, vol. 41, 1913, p. 421.

<sup>g</sup> Gleason, E. D., U. S. Patent 1076973, Oct. 28, 1913.

<sup>h</sup> Mellen, G., U. S. Patent 1092935, Apr. 14, 1914.

<sup>i</sup> Desch, C. H., Metallography, 1910, p. 106.



able protecting layers for low-melting alloys. The melting point of the eutectic mixture ( $2\text{KCl}.\text{LiCl}$ ) of potassium and lithium chlorides is given <sup>a</sup> as  $350^{\circ}\text{C}$ . and that of the lowest melting mixture of  $\text{NaCl}$  and  $\text{LiCl}$  (27 per cent by molecular weight of  $\text{NaCl}$ , 73 per cent  $\text{LiCl}$ ) as  $550^{\circ}\text{C}$ .

Burgess <sup>b</sup> gives the melting point of an equimolecular mixture of  $\text{NaCl}$  and  $\text{KCl}$  as about  $650^{\circ}\text{C}$ .

A flux recently put on the market at 25 cents per pound is said by its introducers to act as a molten cover to protect the metal from the air and also as a flux to dissolve impurities from the melt. The directions state that 1 pound of flux should be used to 250 pounds of metal and scattered over the cold metal as much as possible, so that the metal will be protected while being heated. After all the metal is melted it should be stirred with an iron rod, when the flux will come to the top, as it melts below the melting point of aluminum and rises as soon as the metal is melted. For running down scrap or skimmings, the flux is to be used in the same way, but more must be used than when clean metal is melted, the amount depending on the content of impurities.

Cryolite (sodium aluminum fluoride) is sometimes used as a flux in running down borings, as this is the main constituent of the electrolyte used in the reduction of aluminum, and aluminum oxide is quite soluble in it. The melting point of cryolite (about  $980^{\circ}\text{C}$ .) is high for the purpose, and various substances might be added to lower it.

In the direct testimony of A. H. Cowles before the Circuit Court of the Northern District of Ohio, May 6, 1899, C. M. Hall is quoted as stating that  $\text{KCl}$  and  $\text{NaCl}$  fluxes do not dissolve oxides and other dirt from aluminum, but fluorides do.<sup>c</sup>

Wright <sup>d</sup> states that the electrolyte used in the production of aluminum is composed of cryolite ( $\text{AlF}_3.\text{NaF}$ ), aluminum fluoride, and fluorspar in the proportions of about 16 per cent  $\text{CaF}_2$ , 28 per cent ( $\text{AlF}_3.\text{NaF}$ ), 56 per cent  $\text{AlF}_3$ , the  $\text{AlF}_3$  being made by treating bauxite with  $\text{HF}$ . This is evidently taken from the United States patent 400667 of C. M. Hall, whose various patents Nos. 400665, 400666, 400667, and 400766 cover  $\text{AlF}_3$  plus fluorides of  $\text{Na}$ ,  $\text{K}$ ,  $\text{Li}$ , and  $\text{Ca}$ .

Pascal <sup>e</sup> intimates that cryolite and fluorspar are both used in the electrolyte. He finds that the eutectic mixture of  $\text{CaF}_2$  and  $\text{Al}_2\text{O}_3$

<sup>a</sup> Zemczuzny, S., and Rambach, F., *Schmelzen des alkali Chloride*: Zeit. anorg. chem., Bd. 65, 1910, p. 403.

<sup>b</sup> Burgess, G. K., *The measurement of high temperatures*, 1912, p. 451.

<sup>c</sup> See also Cowles, A. H., *The solubility of alumina in a bath of fused fluorides*: Met. and Chem. Eng., vol. 11, 1913, p. 177.

<sup>d</sup> Wright, J., *Electric furnaces and their industrial applications*, 1908, pp. 186-187.

<sup>e</sup> Pascal, P., *Die Elektrometallurgie des Aluminium*, Das ternäre System Tonerde-Fluorite-Kryolith: Ztschr. Electrochem., Bd. 19, 1913, p. 610. Pascal, P. and Jouniaux, M., *Études physico chimiques sur l'électrometallurgie de l'aluminium*: Rev. met., t. 11, 1914, p. 1069; Bull. soc. chim. France, t. 15, 1914, p. 312; Chem. Abs., vol. 7, 1913, p. 2904, vol. 8, 1914, pp. 2529, 3269.



contains 27 per cent  $\text{Al}_2\text{O}_3$  and melts at  $1,270^\circ\text{C}$ , whereas Lorenz <sup>a</sup> finds that it contains 33 per cent  $\text{Al}_2\text{O}_3$  and melts at  $940^\circ\text{C}$ . Hence fluorspar is a solvent for alumina.

Pascal finds a ternary eutectic for cryolite, fluorspar, and alumina at the proportions 59.3 per cent cryolite, 23 per cent fluorspar, and 17.7 per cent alumina—a mixture which melts at  $868^\circ\text{C}$ .

However, in the published accounts <sup>b</sup> of laboratory experiments for making aluminum, natural cryolite, without the use of  $\text{CaF}_2$  or  $\text{AlF}_3$ , seems to be the electrolyte used.

Neumann and Olsen mention a mixture of 8 parts cryolite to 1 of  $\text{NaCl}$ , and also appear to have added some  $\text{AlF}_3$  to the electrolyte in some tests. The addition of  $\text{NaCl}$  to fluorides in the manufacture of aluminum is also mentioned by Minet.<sup>c</sup> A mixture of 40 parts  $\text{AlF}_3$  and 60 parts  $\text{NaCl}$ , another of 20 to 50 parts cryolite and 80 to 50 parts  $\text{NaCl}$ , and still another of 35 parts  $\text{AlF}_3$ , 10 parts  $\text{NaF}$ , and 55 parts  $\text{NaCl}$  are suggested.

Plato <sup>d</sup> states that a mixture of 60 parts  $\text{KCl}$  and 40 parts  $\text{KF}$  melts at about  $610^\circ\text{C}$ , and a mixture of about 86 parts  $\text{CaCl}_2$  and 14 parts  $\text{CaF}_2$  melts at about  $650^\circ\text{C}$ .

That fluorspar was a flux for  $\text{Al}_2\text{O}_3$  was stated by Mierzinski <sup>e</sup> as far back as 1855, and fluorspar is known to be a good flux for silicates, such as the dirt in most dirty borings doubtless is.

Fedotieff and Iljinsky <sup>f</sup> find that the eutectic mixture for  $\text{NaF}$  and  $\text{AlF}_3$  consists of 36.5 per cent  $\text{NaF}$  and 63.5 per cent  $\text{AlF}_3$  by weight and melts at  $685^\circ\text{C}$ . For the manufacture of aluminum they recommend a mixture of the formula  $5\text{NaF}.\text{AlF}_3$ ; this mixture has a lower specific gravity than the eutectic, but has to be run at  $900^\circ\text{C}$  in order that the metal may separate well. The use of  $\text{CaF}_2$  is not advised, as its specific gravity is even higher than that of aluminum. A mixture consisting of 85 parts  $\text{CaCl}_2$  and 15 parts  $\text{CaF}_2$  that melts at about  $650^\circ$  to  $660^\circ\text{C}$  and has a specific gravity of 2.5, slightly higher than that of molten aluminum, is mentioned by several workers <sup>g</sup> on the electrolytic production of calcium.

<sup>a</sup> Lorenz, R., Jabs, A., and Eitel, W., Beiträge zur Theorie der Aluminiumdarstellung; Ztschr. anorg. Chem., vol. 83, 1913, p. 39.

<sup>b</sup> Richardson, H. K., Some observations on the laboratory production of aluminum: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 159; Thompson, M. DeKay, The electrolytic reduction of aluminum as a laboratory experiment: Electrochem. and Met. Ind., vol. 7, 1909, p. 19; Tucker, S. A., The preparation of aluminum in the laboratory: Electrochem. and Met. Ind., vol. 7, 1909, p. 315; Neumann, B., and Olsen, H., Production of aluminum as a laboratory experiment: Electrochem. and Met. Ind., vol. 8, 1910, p. 185; Haber, F., and Geipert, R., Versuche über Aluminiumdarstellung: Ztschr. Elektrochem. Bd. 8, 1902, p. 1; Borchers, W., Electric smelting and refining, translated by McMillan, W. G., 1904, p. 148.

<sup>c</sup> British Patent 10057, July 18, 1887.

<sup>d</sup> Plato, W., Erstarrungserscheinungen an anorganischen Salzen und Salzgemischen: Ztschr. phys. Chem. Bd. 58, 1907, pp. 362, 364.

<sup>e</sup> Mierzinski, S., Die Fabrikation des Aluminum und der alkaline Metalle, 1855, p. 56.

<sup>f</sup> Fedotieff, P. P., and Iljinsky, W., Beiträge zur Elektrometallurgie des Aluminums: Ztschr. anorg. Chem., vol. 80, 1913, p. 113.

<sup>g</sup> Ruff, O., and Plata, W., Zur Darstellung des Calciums: Ber. deut. chem. Gesell., Jahrg. 85, 1902, p. 3612. Wöhlert, P., Darstellung von metallischen—Calcium für Laboratoriumszwecke; Ztschr. Elektrochem., Bd. 11, 1905, p. 612. Bürgel, C., Über die elektrolytische Gewinnung des Calciums: Ztschr. Elektrochem., Bd. 14, 1907, p. 31.

Arndt and Loewenstein<sup>a</sup> show that molten  $\text{CaCl}_2$  can dissolve 3 per cent  $\text{SiO}_2$  at  $820^\circ \text{C}$ ., 5.5 per cent at  $900^\circ \text{C}$ ., and 7.5 per cent at  $950^\circ \text{C}$ ., and that its specific gravity at  $800^\circ \text{C}$ . is 2.05 and at  $900^\circ \text{C}$ . is 2.00.

Moldenhauer and Anderson<sup>b</sup> give  $630^\circ \text{C}$ . as the melting point of the two lowest melting mixtures of  $\text{CaCl}_2$  and  $\text{KCl}$ , one consisting of 85 parts  $\text{CaCl}_2$  and 15 parts  $\text{KCl}$ , and the other, 40 parts  $\text{CaCl}_2$  and 60 parts  $\text{KCl}$ .

It is therefore seen that there are a number of possible mixtures of alkali and alkaline-earth chlorides and fluorides having comparatively low melting points that might serve as a molten cover or flux in melting aluminum chips. Salts of heavy metals are not promising, on account of the reduction of the salts to metal by aluminum, and the salts of few other acids are sufficiently stable at the necessary temperature.

#### BRIQUETTING BORINGS.

Again it has been suggested that the borings be briquetted, the theory being that in this way the air held by the loose borings is eliminated and the borings put in such shape that they may be readily charged and submerged below the surface of a heel of molten metal.

Such briquets, if made under very heavy pressure, are almost as solid, when cold, as ingot metal. Sperry<sup>c</sup> advocates this treatment, and says—

In the treatment of aluminum chips this process is particularly important as this metal, more than any other commercial one, is difficult to treat in such a form. When briquetted, the melting would become a simple operation and the resulting metal would be worth using. Metal now made from aluminum chips is of the poorest quality.

It is claimed<sup>d</sup> that borings that gave but 50 per cent recovery when melted loose gave 85 per cent when briquetted, whereas Hirsch<sup>e</sup> states that in melting aluminum borings loose borings were melted in 50 minutes per crucible (capacity not stated), with a loss of 13.8 per cent; but briquetted borings lost only 8.1 per cent and melted in 35 minutes.

Two properties of finely divided aluminum must be taken account of in any successful method of melting chips. First, the readiness with which aluminum oxidizes, and second, the difficulty with which tiny globules of molten aluminum coated with a film of oxide or dirt coalesce.

<sup>a</sup> Arndt, K., and Loewenstein, W., Über Lösungen von Kalk und Kieselsäure in geschmolzenem Chlorkalcium; Ztschr. Elektrochem., Bd. 15, 1909, p. 784.

<sup>b</sup> Moldenhauer, W., and Anderson, J., Über die elektrolytische Darstellung von calcium Legierungen und Calcium; Ztschr. Electrochem., Bd. 19, 1913, p. 444.

<sup>c</sup> Sperry, E. S., Briquetting metal chips: Brass World, vol. 7, 1911, p. 41.

<sup>d</sup> Anon., Briquetting metal turnings and borings: Engineering, vol. 94, 1912, p. 737; Jour. Inst. Metals, vol. 9, 1913, p. 246.

<sup>e</sup> Hirsch, E. F., Metall-briketts: Elektrotech. Ztschr. vol. 35, 1914, p. 1093.

## OXIDATION OF ALUMINUM.

Magnesium powder is the main constituent of many photographic flash-light powders; aluminum powder is also largely used for this purpose and, when suitably heated in the air, burns with explosive violence. Because of its rapid oxidation, finely divided aluminum is used in illuminating bombs and range-finding shells for night use in warfare. The affinity of aluminum for oxygen is so great that aluminum powder, when mixed with the finely divided oxide of the metal it is desired to reduce and the mixture heated in one spot by a suitable primer, reacts with great violence, forming molten aluminum oxide and the metal desired. This is the well-known thermit reaction so much used for the production of difficultly reducible metals, and which produces probably the highest temperatures attainable by man, save those of certain electric furnaces.

Finely divided aluminum, mixed with an oxidizing agent such as ammonium nitrate, whose decomposition products are gaseous, has been suggested as an explosive for miners' use.<sup>a</sup>

Aluminum oxide is not reducible to the metal by carbon, even in the electric furnace, only a carbide being formed save in the presence of another metal, as in the Cowles process for the manufacture of aluminum bronze. To reduce the oxide, one must use electrolysis in a fused electrolyte. Aluminum chloride is reducible by metallic sodium or potassium.

Echevarri<sup>b</sup> states that in ordinary melting of ingot the metal should not be heated above 725° C. (1,330° F.), as at that temperature aluminum readily oxidizes with the oxygen of the air.

## NITRIDATION OF ALUMINUM.

Not only does the oxygen of the air but also the relatively inert nitrogen combine with finely divided, highly heated aluminum. Several workers<sup>c</sup> find that if aluminum powder is heated in a current of nitrogen to a temperature of about 700° C. the temperature rapidly rises, even to 1,300° C. in the interior of the mass. Matignon<sup>d</sup> states that a temperature of even 2,000° C. is reached, with the formation of aluminum nitride, which coats the tiny globules so that they do not coalesce.

Zappi<sup>e</sup> has noted that the more finely divided metallic aluminum is, the greater the protective effect of the oxide coat, as large pieces

<sup>a</sup> Sesti, G., Italian Patent 133882, Aug. 7, 1913; Chem. Abs., vol. 1, 1915, p. 1285.

<sup>b</sup> Echevarri, J. T. W., Aluminum and some of its metals used: Jour. Inst. Metals, vol. 1, 1909, p. 128.

<sup>c</sup> Shokoff, I., Aluminum nitride: Jour. Russ. Phys. Chem. Soc., vol. 40, 1908, p. 457; Sci. Abs., vol. 13, 1910, p. 517; Wolf, J., Zur Darstellung von Aluminiumnitride aus den elementen: Ztschr. anorg. Chem., Bd. 83, 1913, p. 159; Fichter, F., Über Aluminiumnitride: Ztschr. anorg. Chem., Bd. 54, 1907, p. 324; Fichter, F., and Spengel, A., Die Reactionen des Aluminiumnitride: Ztschr. anorg. Chem., Bd. 82, 1913, p. 192.

<sup>d</sup> Matignon, C., Synthesis of ammonia by means of aluminum nitride: Jour. Franklin Inst., vol. 178, 1914, p. 794.

<sup>e</sup> Zappi, E. V., Action of metals on chloride of carbon: Anales soc. quim. Argentina, vol. 2, 1914, p. 217; Chem. Abs., vol. 9, 1915, p. 2117.



of metallic aluminum react with carbon tetrachloride at 100° C., while aluminum powder does not react at 280° C.

Vickers <sup>a</sup> states that in melting borings under a heel of molten metal considerable nitrogen is absorbed, which will produce minute pinholes in the castings. He states that 0.5 per cent nitrogen will reduce the strength of aluminum alloy one-half, that it is practically impossible to prevent the absorption of nitrogen in melting turnings, and that the only way to remove the nitrogen is to add titanium. Rossi <sup>b</sup> claims that the addition of titanium to aluminum removes both nitrogen and carbon as titanium nitride and cyanonitride.

Sperry <sup>c</sup> states that aluminum absorbs nitrogen while melting. It is also reported <sup>d</sup> that the higher the temperature to which the metal is heated the more nitrogen is taken up. On the other hand, Fichter <sup>e</sup> states that no nitride is formed in melting large masses of aluminum, even if nitrogen be blown into the melt.

In unpublished experiments of H. S. Bennett, of Cornell University, nitrogen was bubbled through 7 pounds of molten pure aluminum held at 960° C. to 1,010° C. for 70 minutes. Physical tests (including the "hot shortness" test <sup>f</sup>) showed no change in properties, although analysis indicated the presence of traces of nitrogen.

Carpenter and Edwards <sup>g</sup> found a nitrogen content ranging from nothing up to 0.0055 per cent in some of the aluminum used by them. The chief chemist of a large aluminum company states that he has never found over 0.001 per cent. One laboratory states that aluminum may contain from 0.0024 to 0.06 per cent. On two samples of a lot of new ingot metal suspected of containing nitrogen, one laboratory found 0.0571 and 0.0376 per cent, whereas a very careful analyst working on duplicate samples of this lot found only 0.0007 and 0.0004 per cent, and in another lot known to have been very strongly overheated he found 0.0004 per cent.

Inasmuch as aluminum nitride does not appear to be soluble in metallic aluminum, it seems probable that if any nitride really exists in ingot refined from borings, it is mechanically included, although Tchijerski <sup>h</sup> claims that aluminum nitride can form a solid solution with iron. However, nitrogen is absorbed in melting borings or in remelting skimmings, when finely divided metal is present and the mass is heated very hot. A nitrogen content that would correspond

<sup>a</sup> Vickers, C., How to melt aluminum turnings and borings: Foundry, vol. 41, 1913, p. 119.

<sup>b</sup> Rossi, A. J., U. S. Patents 1,056,125, 1,085,488, 1,104,371.

<sup>c</sup> Sperry, E. S., Notes: Brass World, vol. 8, 1912, pp. 134, 270.

<sup>d</sup> Anon: Some hints on the melting of aluminum: Castings, vol. 11, 1911, p. 2.

<sup>e</sup> Fichter, F., Über aluminiumnitride: Ztschr. anorg. Chem., Bd. 54, 1907, p. 324.

<sup>f</sup> Norton, A. B., A "hot shortness" testing machine for aluminum alloys: Trans. Am. Inst. Metals, vol. 8, 1914, pp. 124-127.

<sup>g</sup> Carpenter, H. C. H., and Edwards, C. A., Eighth report to the Alloys Research Committee: Proc. Inst. Mech. Eng., 1907, p. 57.

<sup>h</sup> Tchijerski, N. P., Iron and nitrogen: Rev. Soc. Russ. Met., vol. 1, 1913, p. 127; Science Abs., vol. 13, 1915, p. 256; Rev. met., t. 11, 1915, p. 617.



to 15 per cent AlN has been found in the dross taken off in the remelting of skimmings, a little  $\text{ZnCl}_2$  being used as flux.

Richards <sup>a</sup> has called attention to the fact that the dross skimmed off of molten aluminum gives off ammonia in moist air, owing to the formation of aluminum nitride.

On the absorption of nitrogen by mixtures of alumina and carbon at high temperatures are based several processes for the fixation of atmospheric nitrogen, such as those of Serpek, who aims to produce aluminum nitride, and of Peacock, who aims to produce a carbide. Whatever be the exact product or the reaction that takes place in refining borings, it is certain that both the oxygen and nitrogen of the air enter into the formation of the dross. It is also stated <sup>b</sup> that in melting aluminum in graphite crucibles, if the metal is allowed to become very hot and is then stirred so that particles of graphite are scraped off the crucible, some aluminum carbide will be formed. Whether this be true or not, there is some indication that minute traces of some carbide, whether a carbide of aluminum or of some impurity is not known, may be present in most aluminum, as the odor of a freshly fractured surface, such as that of a tensile test bar, often decidedly reminds one of a faint acetylene-like odor <sup>c</sup> and seems to be stronger on the fracture of metal obtained by remelting borings than on that of ordinary ingot. As aluminum carbide ( $\text{Al}_4\text{C}_3$ ) with water produces methane instead of acetylene, it is peculiar that so marked an odor should be present if the carbide is one of aluminum itself.

#### COALESCENCE OF MOLTEN ALUMINUM GLOBULES.

Whether the globules formed when finely divided aluminum is melted are covered with a thin film of oxide, or mixed oxide and nitride, or possibly carbide, or with fine dirt, they do not coalesce readily, but remain as distinct globules, something like drops of water on a dusty table. If a mass of these coated globules could be pictured the mass would show a honeycomb-like formation with the drops of molten metal as the honey and the film of dirt and oxide as the comb. In other words, there is an emulsion of a solid (the film) and a liquid (the metal). If this emulsion is not entirely broken up and the metal freed from the enveloping film, the whole mass of oxide and dirt with the inclosed metal will be skimmed off from what metal has coalesced in the bottom of the crucible. As soon as air strikes this mass of hot porous dross, as it does when the dross is skimmed off, the tiny metallic globules entangled in the dross oxidize at once. The heat of the reaction is so great that a layer of such

<sup>a</sup> Richards, J. W., Aluminum nitride: Trans. Am. Electrochem. Soc., vol. 23, 1913, p. 351.

<sup>b</sup> Anon., Aluminum melting difficulties: Foundry, vol. 42, 1914, p. 73.

<sup>c</sup> See Beckman, J. W., In discussion: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 177.

dross a few inches thick put on an iron plate a quarter of an inch thick will rapidly burn a hole through the iron.

Fairly large borings free from dirt form comparatively large globules, which can by their own weight break through the enveloping film of oxide, coalesce well, and melt down without much loss. But the smaller borings, that will pass a 20-mesh sieve and may have a thickness of but 0.005 inch or less, form, it will be seen, almost microscopic globules when melted, the weight of which is not sufficient to break through even the very thin film of oxide, to say nothing of adhering dirt.

Skinner and Chubb<sup>a</sup> state that the oxide film obtained in the electrolytic insulation of aluminum wire which is from 0.0001 to 0.0004 inch thick will stand up without rupturing, in small coils, even when the coil is carrying so much current that the wire is molten within this extremely thin shell.

Seligman<sup>b</sup> and Morrison<sup>c</sup> as well as Schoop<sup>d</sup> state that the film of oxide on untreated aluminum wire, which is, of course, infinitely thinner than the film on the electrolytically insulated wire, prevents two wires from welding together without the use of a flux to dissolve the oxide and give true metallic contact.

In "calorizing" iron or steel—that is, coating it with a film of aluminum and aluminum oxide—the piece to be calorized is packed in a mixture of 5 to 50 parts powdered aluminum, 94 to 49 parts aluminum oxide, and 1 part ammonium chloride and then heated to 900° to 950° C.,<sup>e</sup> or far above the melting point of the aluminum. The globules of aluminum do not coalesce.

## ANALOGOUS PROBLEMS IN COALESCENCE OF OTHER METALS.

### MERCURY AND TIN.

The failure of the aluminum to coalesce is similar to the behavior of floured or sickened mercury, globules of which when coated with talc, graphite, grease, or other materials of that nature, coalesce with the greatest difficulty.<sup>f</sup> Another case where coalescence of particles of a metal is inhibited by a network of foreign material about them is in fusible tin boiler plugs, where Burgess and Merica<sup>g</sup> find that a little tin oxide may prevent the blowing out of a boiler plug even after the tin is all melted.

<sup>a</sup> Skinner, C. E., and Chubb, L. W., The electrolytic insulation of aluminum wire: *Trans. Am. Electrochem. Soc.*, vol. 26, 1914, p. 137.

<sup>b</sup> Seligman, R., The welding of aluminum: *Jour. Inst. Metals, British*, vol. 2, 1909, p. 281.

<sup>c</sup> Morrison, W. M., Discussion: *Jour. Inst. Metals, British*, vol. 1, 1909, p. 150.

<sup>d</sup> Schoop, M. V., Autogenous welding of aluminum: *Electrochem. Met. Ind.*, vol. 7, 1909, p. 151.

<sup>e</sup> Ruder, W. E., Calorizing metals: *Trans. Am. Electrochem. Soc.*, vol. 27, 1915, p. 254.

<sup>f</sup> See Gowland, W., The metallurgy of the nonferrous metals, 1914, p. 305.

<sup>g</sup> Burgess, G. K., and Merica, P. D., An investigation of fusible tin boiler plugs; *Jour. Wash. Acad. Sci.*, vol. 5, 1915, p. 461. *Trans. Am. Inst. Metals*, vol. 9, 1915. Adv. copy 3 pp.

## BLUE POWDER.

A closer analogy, however, is that of the difficultly meltable "blue powder" obtained in zinc smelting. Blue powder, according to the general consensus of opinion, consists of tiny globules of zinc which, as they condense from the retort, become coated with a thin film consisting mainly of zinc oxide with perhaps some silica.

The presence of an oxide coating on the particles of blue powder is an advantage for one particular purpose, that of sherardizing, since this coating prevents the coalescence of the zinc dust, which is in this case undesirable. Hence, according to Trood,<sup>a</sup> zinc oxide in a zinc dust for sherardizing purposes should not be below 8 per cent.

Roeber<sup>b</sup> distinguishes between physical blue powder, or zinc snow, which is finely divided pure zinc free from any coating and according to Hansen<sup>c</sup> can be easily remelted, and chemical blue powder, which is finely divided zinc coated with oxide and silica and can not be remelted without great loss.

Ingalls,<sup>d</sup> Johnson,<sup>e</sup> Lyon and Keeney,<sup>f</sup> and West,<sup>g</sup> explain the formation of unmeltable blue powder in similar ways.

Peterson<sup>h</sup> states, however, that blue powder has the same chemical analysis as spelter, the difference being merely a physical one, and that attempts to melt blue powder in open vessels fail because of oxidation during the heating, before consolidation can take place.

Bleeker,<sup>i</sup> on the other hand, states that blue powder contains but 80 to 95 per cent of metallic zinc, the rest being oxide, with traces of other impurities as well.

Ingalls<sup>j</sup> gives analyses showing the presence of 7.5 to 10 per cent of zinc oxide, on the average.

By putting blue powder into fused zinc chloride held at a temperature above the melting point of zinc, Bleeker found that some of the zinc coalesced, and that on passing a direct electric current through the molten bath the suspended zinc particles migrated to the cathode and coalesced. The recovery was over 100 per cent, owing to electrolysis of the zinc chloride.

Peterson<sup>k</sup> got promising results by this method. He also tried putting blue powder in an iron cylinder with notches, corresponding

<sup>a</sup> Trood, S., Sherardizing: Trans. Am. Inst. Metals, vol. 9, 1915. Advance copy, 11 pp.

<sup>b</sup> Roeber, E. F., Editorial, Met. and Chem. Eng., vol. 10, 1912, p. 451.

<sup>c</sup> Hansen C. A., In discussion, Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 138.

<sup>d</sup> Ingalls, W. R., The electric smelting of zinc ore: Met. and Chem. Eng., vol. 10, 1912, p. 482.

<sup>e</sup> Johnson, W. McA., The art of electric zinc smelting: Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 191. Met. and Chem. Eng., vol. 13, 1915, p. 763; U. S. Patent 1,150,271, Aug. 17, 1915.

<sup>f</sup> Lyon, D. A. and Keeney, R. M., Possible applications of the electric furnace to western metallurgy: Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 138.

<sup>g</sup> West, T., Determination of oxygen in copper and brass: Jour. Inst. Met., vol. 10, 1913, p. 379.

<sup>h</sup> Peterson, P. E., The electric zinc furnace: Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 231.

<sup>i</sup> Bleeker, W. F., An electrolytic method for the reduction of blue powder: Trans. Am. Electrochem. Soc., vol. 21, 1912, p. 359.

<sup>j</sup> Ingalls, W. R., The metallurgy of zinc and cadmium, 1903 ed., pp. 205, 529.

<sup>k</sup> Peterson, P. E., Loc. cit.



to tap holes, in the bottom, placing an iron plunger actuated by a screw on top of the powder, heating the whole to  $600^{\circ}$  to  $700^{\circ}$  C., and applying pressure with the plunger. In this way he was able to break the film of oxide and make the zinc globules coalesce, getting a recovery of 80 to 85 per cent.

Peterson's furnace is merely a modification of the old Montefiore furnace for the melting of blue powder, described by Ingalls.<sup>a</sup> This furnace contained upright boot-shaped retorts into which 40 to 50 pounds of blue powder was charged, a clay piston weighted with an iron bar being put on top of the powder in the leg of the boot to supply pressure to break the oxide film. The metal ran out from the toe of the boot. The yield was 85 per cent to 86 per cent, but the metal occluded a good deal of oxide and was hence of poor quality, so that the use of the furnaces was generally given up in favor of redistilling the blue powder.

Cote and Pierron<sup>b</sup> suggest a continuous furnace for the melting of blue powder, in which the powder is fed into a horizontal cylinder within which rotates a solid eccentric cylinder, smooth on one side and bearing helically arranged flanges on the other. The smooth side almost or quite touches the interior of the hollow cylinder and exerts pressure on the mass of blue powder.

The flanges act as a screw conveyor to move the material from the hopper opening at one end to the discharge opening at the other. The blue powder is mixed with a suitable flux and fed into the hollow cylinder, which is kept at  $450^{\circ}$  to  $500^{\circ}$  C. ( $30^{\circ}$  to  $80^{\circ}$  above the melting point of zinc) by various means, among which electrical heating is mentioned, triturated by the inner cylinder and agglomerated into semifluid masses in which the tiny globules have coalesced into much larger ones, and which may therefore be remelted and the oxide released.

It is stated in the patent that from blue powder containing 85 per cent of metallic zinc 80 per cent of the metal, or 68 per cent of the original blue powder, may be recovered, even in a "poor apparatus," and that under good conditions the whole of the metal content may be recovered. As fluxes, sodium, potassium, and calcium chlorides, carnallite, zinc chloride, and sodium and potassium chlorozincates, or mixtures of these, are mentioned. A flux consisting of 60 per cent of NaCl and 40 per cent  $\text{CaCl}_2$ , used in the proportion of 15 to 20 per cent of the weight of the blue powder, is recommended.

No information is at hand as to whether or not this furnace has proved commercially successful or as to the maintenance cost.

Richards<sup>c</sup> states that one of the difficulties in the electrolytic refining of zinc is the deposition of spongy zinc, which loses 20 to 25 per cent on remelting, though this loss is decreased by using sal ammoniac as a flux.

<sup>a</sup> Ingalls, W. R., *The metallurgy of zinc and cadmium*, 1903 ed., p. 527.

<sup>b</sup> Cote, E. F., and Pierron, P. R., French Patent 458111, July 29, 1912.

<sup>c</sup> Richards, J. W., *The electrolytic deposition of zinc*: Trans. Am. Electrochem. Soc., vol. 25, 1914, p. 281.



Thierry <sup>a</sup> states that attempts made to melt blue powder with sal ammoniac result in a loss of zinc and present numerous difficulties so that the method had not been used in commercial practice.

He further states that in order to make the powder coalesce on melting it is desirable to remove the oxide coating by adding just sufficient hydrochloric acid to dissolve the oxide, without leaving any excess acid to act on the metallic zinc, the proper amount of acid being determined by analysis. The cleaned powder is then compressed to remove the solution, and the blocks melted down out of contact with air. Information as to the commercial operation of this process is lacking.

#### SUMMARY OF METHODS SUGGESTED.

The methods suggested in the literature for melting aluminum chips and for the analagous problem of melting "blue powder" fall into the following general classes:

1. Feeding the chips back into the aluminum reduction pots and subjecting the melt to electrolysis.

2. The use of a flux, or, more strictly, a molten cover, melting at or below the melting point of aluminum or of the alloy to be remelted, in order to exclude air in ordinary furnaces.

3. The use of a true flux that will dissolve aluminum oxide and dirt.

4. Exclusion of air by stirring the chips into a previously molten "heel" of aluminum in an ordinary furnace.

5. Exclusion of air by the use of a vacuum furnace or a retort furnace, or by utilizing the stored heat in the walls of a previously heated furnace.

6. Briquetting the chips while cold in order to press adjacent chips closely together and thus make coalescence more easy, to reduce the amount of air held by the borings, and to permit easy poking of the borings down under the surface of the molten metal.

7. Subjecting a mass of chips to pressure while melting to promote coalescence, or subjecting it both to slight pressure and to constant stirring, as in the puddling method.

8. Promoting coalescence by stirring a volatile material such as  $\text{NH}_4\text{Cl}$ ,  $\text{ZnCl}_2$ , or  $\text{AlCl}_3$ , which vaporize at  $350^\circ$ ,  $730^\circ$ , and  $183^\circ \text{C.}$ , respectively, into the mass of globules of metal mixed with oxide and dirt. The gas evolved lifts the particles and stirs them so that some of the tiny globules may come into metallic contact and coalesce. There is also a possibility that the nascent chlorine formed by the dissociation of the substances may react with the oxide film to form the chloride and thus clean the surface of the globule.

9. Cleaning the surface of the chips with chemical solutions before melting.

---

<sup>a</sup> Thierry, C. V., U. S. Patent 1030351, June 25, 1912.

**METHODS USED COMMERCIALY IN MELTING CHIPS.**

Seligman <sup>a</sup> states that in British practice, as a rule, no flux is used in remelting chips and that the loss is high because of the high temperature needed to free the metal from the dross. He does not consider the use of  $\text{NH}_4\text{Cl}$  or  $\text{ZnCl}_2$  to be of much value and states that the use of  $\text{ZnCl}_2$ , particularly with pure aluminum, is disadvantageous, as it contaminates the metal with zinc. He has obtained nearly perfect recovery by adding fine scrap and turnings to the reduction pots, but the impurities are likely to upset the bath. He has had good recoveries when using low-melting fluoride-containing fluxes similar to the Schoop flux for welding, even getting ingot from aluminum powder, but such fluxes attack the crucible.

**PRACTICE AT AN AMERICAN PLANT.****CRUCIBLE METHOD.**

In the practice of one American refining company borings have been melted in various ways, first in coke, pit furnaces, with a small amount of common salt as flux, and later with No. 250 crucibles in either coke or oil-fired pit furnaces. The crucibles were charged with borings and set into the furnace. When the borings became pasty—that is, when the metal melted but had not yet broken through the skin of oxide plus dirt that had formed about each chip—the mass was poked down with a stirrer and more borings added through a long sheet-iron funnel. This was repeated until the crucible was full of molten metal and dross. A little sal ammoniac (commercial ammonium chloride) was added from time to time as the metal was poked and stirred. If the dross seemed free from metallic particles, some of it was taken off with a skimmer during the process of melting, so as to give more room in the crucible. It was considered that the more often the metal was poked down, so as to break the oxide film inclosing the particles, the better would be the results. At the end of the melt the pot was pulled from the fire, and more sal ammoniac was added and stirred and poked well into the dross, a total weight of sal ammoniac equal to about 1 per cent of the weight of borings used being the usual quantity added.

The dross skimmed off in this method was usually somewhat caked together, still contained small globules of molten metal, and was very hot. As soon as it was exposed to the air, violent oxidation, much like the thermit reaction, set in, and the mass glowed with an almost blinding light. If the dross was not spread out thinly, so as to cool rapidly, or else covered thickly with sand, so as to exclude the air, it would get hotter and hotter, evidently from the oxidation and nitridation of the metallic aluminum still con-

---

<sup>a</sup> Seligman, R., in personal letter to G. M. James, March, 1915.

tained in it. When a pile of this dross a few inches thick was put on a  $\frac{1}{4}$ -inch iron plate, it would melt a hole through the iron in a short time, and the pile would glow for hours. The dross, when put into water or allowed to stand in moist air gave off ammonia.

After the dross had been skimmed from the surface of the metal and any adhering below the surface to the walls or bottom of the crucible had been loosened and skimmed off, the metal was cooled by adding some solid ingot poured from a previous melt, as it was so hot that if poured without cooling the ingot would have a poor surface appearance.

It was necessary to get the metal very hot, as otherwise even the larger globules of metal held by the dross would not be fluid enough to break through the enveloping dross and coalesce.

#### MELTING IN IRON POTS, WITH PUDDLING AND ZINC CHLORIDE FLUX.

This firm has lately changed from crucible melting to iron-pot melting, the new furnaces being similar to those commonly used in aluminum foundries, where it is now more common to melt aluminum alloys in cast-iron pots than in crucibles. The net metal loss in melting ingots for casting in iron pots is not over one-third of 1 per cent, if reasonable care is taken, when the recovery of metal made on remelting the skimmings from the first melting is taken account of.

The furnace used is cylindrical, lined with fire brick, and covered by a heavy iron plate with a hole in it, into which the pot sets, a flange at the edge of the pot holding it suspended in the furnace. The furnace is not sunk in a pit, but stands about  $3\frac{1}{2}$  feet high above the floor. The furnace is fired with oil, the flame from the burner entering the combustion chamber below the pot through a hole in the furnace shell and the waste gases being taken off through a port in the shell near the top. The furnace is usually not tilting, and the metal is ladled out, but tilting furnaces using iron pots have recently come into use for melting ingot. In such furnaces the products of combustion do not sweep over the surface of the metal, as they do in a pit furnace, and the metal loss in melting ingot aluminum alloy is less than in crucibles in a coke fire. Resting on the top of the furnace is a sheet-iron cylinder, with a wide door in the front leading into a conical hood, which takes the fumes from the cutting oil and other impurities and the volatile fluxes off to a short sheet-iron stack.

A few borings are put into the pot and heated, with constant stirring, until they become pasty or semifluid. More borings are then added to the mass, which is not allowed to become really molten, but is kept mushy. The borings are puddled all the time, a furnace tender being required for each furnace. The flame, the stirring,



and the feeding of borings are all so regulated that at no time is the metal allowed to become even faintly red.

If the flame is run too high, if the stirring is not sufficient, or if the borings are fed in too fast for proper stirring, oxidation will start in the mass and the temperature will rise rapidly, with consequent increased oxidation and further rise in temperature, till nearly the whole mass will be converted to oxide. Hence the charge is kept small enough—probably between 100 and 150 pounds—so that the furnace tender can keep it all constantly worked over. When no more borings can be added and efficiently stirred in, the pot is covered and the temperature raised somewhat, so as to allow the metal to settle and the dross to rise. Then the bottom and sides of the pot are scraped free from dross and about 2 per cent of fused  $\text{ZnCl}_2$  is stirred vigorously into the dross on top. After the zinc chloride has volatilized the dross is taken off with a perforated skimmer. Each skimmerful is allowed to remain in the air only the shortest possible time, the dross being at once dropped into an iron tank partly full of water. Much ammonia is evolved as the nitride-bearing dross reaches the water, and some inflammable gases, probably hydrocarbons, are also produced. With this type of furnace it is possible to stir the metal continuously, which was not the case with crucibles in pit furnaces. The vital points of the process lie in keeping the temperature down while the borings are being added and in constant stirring to promote coalescence.

It is difficult to poke borings down into a crucible or to stir them in a pit fire because of the heat, the necessary length of the stirrer handle, and the choking fumes that arise from the oily borings and from the volatile fluxes used at the end of the heat. In the pot method the heat is not great, the workman is only a couple of feet from the pot and can use his stirrer to better advantage, and the fumes are pretty well taken off.

By the pit-furnace method this firm found it difficult to obtain over 50 per cent recovery, whereas with the pot method the recoveries on the original material, not on the metal content, are given as follows: For 1913, 67.8 per cent, for 1914, 68 per cent, and for the first part of 1915, 69.6 per cent. These figures are on material which includes a certain proportion of centrifuged borings. These borings are sent to the refining plant by an automobile company that uses a water-soluble cutting compound, and states that it uses a centrifugal dryer running at 600 revolutions per minute for drying aluminum chips. The dryer takes a charge of about 175 pounds and three or four minutes whirling reduces the liquid content in the borings to 3 to 5 per cent. No water is added to wash out the cutting compound. After the borings are centrifuged they are stored in sheds until a carload has accumulated, when they are sent to the refiner. The average



recovery on the dried borings is stated by the company to be 85 to 86 per cent, whereas previous to such drying the recovery on the borings was 50 to 60 per cent. There is little dirt in these borings, as the floors about the machines are kept well scrubbed.

Before the centrifuge was used for drying the borings, if borings wet with the cutting compound were placed in the storage bins after a little time fumes would arise that had an odor similar to ammonia or smoked herring, then the borings would corrode rapidly. Now, there is no loss from corroding or overheated borings. Although the recovery is given by the automobile company as 85 to 86 per cent on the borings as shipped from the plant to the refiner, the refiner states that there is usually 4 to 5 per cent of iron borings mixed with the aluminum. The iron is removed as completely as possible by passing the borings over a magnetic separator. Therefore the recovery on the actual aluminum content of the chips is about 90 per cent.

The statement that an odor as of ammonia or smoked herring arose from the damp borings on standing in the air, heated only by the heat of reaction due to oxidation, indicated an interesting fixation of atmospheric nitrogen at low temperatures. Some of the cutting compound used was obtained and both clean millings of pure aluminum and oily No. 12 alloy chips were soaked with it and allowed to stand in the air, and also were heated somewhat by external means, but no odor of ammonia or amines was noted.

#### MELTING WITH $\text{ZnCl}_2$ - $\text{NaCl}$ FLUX IN CRUCIBLES.

Another refining plant, which deals but seldom with aluminum chips, reported that it had sprinkled borings with 2 per cent of a strong  $\text{ZnCl}_2$  solution and added 2 per cent of dry common salt, and run the borings down in crucibles in coke fires, getting a recovery of 80 to 82 per cent, but the borings were very clean and nearly free from oil or dirt.

Another refining company which uses a reverberatory furnace is said to get a recovery of 80 per cent. The process is held secret and no details are available as to the exact method of melting, the fluxes used, if any, or as to the quality of borings on which an 80 per cent recovery is made.

#### REMELTING DROSS.

One aluminum foundry which used merely to remelt the skimmings taken from the regular melting pots and to sell the dross from the remelting at \$20 per ton now washes them in a wet tumbling barrel, obtaining a concentrate of 800 to 900 pounds of large buttons, plus the adhering dross, from a ton of original dross. The concentrate is remelted promptly—as if left in the air it would oxidize and heat—and 400 to 500 pounds of ingot, worth \$60 to \$250, depending on the price of aluminum, is obtained, the saving being far greater than

the cost of treatment. The melting is done in iron pots and sal ammoniac is used as flux. The average recovery is only about 60 per cent of the actual metal content of the concentrate.

In melting down aluminum grindings from the emery wheels used in grinding the rough castings, a recovery of about 6 per cent on the total weight of aluminum plus dirt (the dirt forming at least half the weight) was obtained by this firm, using  $\text{ZnCl}_2$  as flux. The grindings as they come from the wheels ordinarily sell for one-half to 1 cent per pound.

This firm made a few experiments on melting borings by heating an iron pot very hot, putting  $\text{ZnCl}_2$  in the bottom and immediately charging all the borings the pot would hold, then heating very rapidly, the idea being to melt rapidly and to stir the whole mass by the rising vapor from the  $\text{ZnCl}_2$ . Definite figures on this method are not available, but it is stated not to have been very satisfactory.

#### MELTING WITH SALT AND FLUORSPAR IN CRUCIBLE COKE FURNACE.

At another large refining plant the borings are passed through a magnetic separator and then over an inclined, vibrating, fine-meshed Newago screen, the screen being continually struck by a set of automatically operated small hammers to keep the openings from clogging. The borings are purchased by weight after as much as possible of the iron and dirt is thus removed. This firm has also experimented with various jigs, shakers, and other mechanical devices for separating brass borings from the aluminum borings in the dry way by means of the difference in specific gravities, and with liquation devices for removing pieces of babbitt and white metal, apparently without very satisfactory results.

Passing the chips over a cloth through which air is blown from below, in order to blow out the fine dirt, has also been tried by this firm, but is apparently not so satisfactory as the screen.

The borings are intimately mixed with quite large proportions, apparently between 15 and 30 per cent, of a flux consisting of common salt with a small percentage, probably about 15 per cent, of fluorspar. The mixture of borings and flux is charged into crucibles of about No. 120 size and heated in coke fires. More mixture is added as soon as the first charge can be packed down to make room for it, until the crucible is full. Little stirring, aside from this packing, is done till the end of the heat. The temperature is raised till the flux is fluid enough for the metal globules to separate from it well, when the melt is stirred to promote coalescence of the globules, skimmed, and poured. The flux and dross skimmed off are allowed to cool, crushed, leached with water, and any metallic buttons are recovered by various means of mechanical concentration. The chemist of this firm states that he considers the problem of melting borings as strictly analagous

to welding, for which a flux that will actually dissolve aluminum oxide and dirt, such as the fluxes made up of alkali chlorides and fluorides, are essential. He chose a salt and fluorspar flux for commercial use because of its cheapness. On perfectly clean borings, of good size, recoveries of 95 to 98 per cent are reported by this method, but it is stated that with very oily and dirty borings, where a good deal of fine dirt, such as floor sweepings, foundry sand, or road dust, the latter sometimes intentionally added by unscrupulous junk dealers, adheres to the borings, the recovery falls to 65 per cent.

This firm occasionally uses the same flux, but melts in a reverberatory furnace heated very hot before being charged, as much of the metal as possible being melted by the heat stored in the walls, but ordinarily prefers the crucible method, although the recovery of metal from the reverberatory process is said to be almost if not quite as good as from crucibles.

### EXPERIMENTS IN MELTING ALUMINUM CHIPS.

#### GENERAL METHOD OF MAKING TESTS.

In order to compare various methods of melting on the same lot of chips, two barrels of "No. 12" (92 per cent Al, 8 per cent Cu) chips were obtained. These chips were from the milling machines and drills of a large automobile manufacturing company, and were thought by the firm supplying them to be of such quality that on commercial melting in iron pots, by the puddling method, the recovery would be about 50 to 65 per cent, as that was the recovery usually obtained by a refining company using that method, on the regular run of borings from this automobile firm.

The tests were necessarily on a small scale, a charge of 1 to 2 kilograms being used, in a small crucible heated in a gas-fired furnace.

The metal was poured into small ingot, the relative weight of which to the original weight of borings is recorded as the "primary" yield in the tables of results. The dross, when possible, was skimmed into water, but if the nature of the flux made this procedure dangerous, the dross was allowed to cool, was crushed, and washed in water. The wet dross, with the admixed buttons of metal, was washed in a stream of water on a 20-mesh riddle. The buttons, if large and clean, were weighed without remelting; if small and with adhering dross, they were remelted with the same flux and in the same manner as the original melt. The large, clean buttons, or the ingot recovered from the smaller buttons, form the "secondary" yield. The sum of the primary and secondary yields is the total yield on the basis of original weight, that is, on the weight of borings plus oil and dirt. The total yield on the basis of metal content is obtained by dividing the total yield on the basis of original weight by the metal content of the dirty borings.



## KIND OF CHIPS USED.

The chips in barrel A contained about 3 per cent of oil and over 7 per cent of very fine dirt. After the oil had been extracted with gasoline the material was screened; 72 per cent remained on a 20-mesh screen, 18 per cent was between 20 and 80 mesh, and 7 per cent passed the 80-mesh screen. Microscopic examination of the material passing the 80-mesh screen showed only traces of aluminum, the vast bulk of it being merely dirt, mainly siliceous, and appeared to be sweepings from a cement floor. Sticks, paper, peach pits, oily waste, chewing tobacco, and apple cores were present in both barrels. All of the dirt could not be sifted out, even after the oil had been extracted with gasoline. The metal content of barrel A was something less than 90 per cent, but 90 per cent is taken in calculating the total yield on the metallic basis. On material from which the oil had not been extracted, so that the fine borings stuck to the larger ones, sieve tests of borings from barrel A gave the following results:

Over 20 mesh, 80 per cent, which equals 76 — per cent of metal plus 4 + per cent of oil and dirt; passing 20 mesh, 20 per cent, which equals 14 — per cent of metal plus 6 + per cent of oil and dirt, figured on weight of original material, or the oily borings over 20 mesh contained about 95 per cent of metal, whereas those passing the 20-mesh screen contained about 70 per cent of metal.

When borings from lot A from which the oil had not been extracted were sized on 14 and 40-mesh riddles the results were: 54 per cent over 14 mesh, 42 per cent between 14 and 40 mesh, 4 per cent under 40 mesh.

The chips in barrel B contained about 2 per cent oil, and on chips from which the oil had not been extracted a sieve test showed the following results: 57.5 per cent over 14 mesh, 30.3 per cent between 14 and 40 mesh, 12.2 per cent under 40 mesh.

The material under 40 mesh was practically all dirt, only traces of metallic aluminum being found in it. As about 5 per cent of oil and dirt could be separated from the material over 40 mesh in size, by extraction with gasoline and sifting on an 80-mesh sieve, the total nonmetal content was about 17 per cent. The metal content of barrel B, for purposes of computation, is taken as 85 per cent. The contents of both barrels had been passed through a magnetic separator, but the removal of iron was far from complete.

## CONTROL OF TEMPERATURE.

Control of the temperature was attempted, an iron-constantin thermocouple of the rod and tube type being used, but the temperatures were found to vary throughout the charge. The upper part of the charge would become much hotter than the lower part when



any oxidation was taking place in the layer of dross and globules of metal overlying the coalesced mass of metal, as when the furnace cover was removed just before taking out the crucible and pouring.

With fluxes that prevented any visible oxidation, the temperature at the bottom of the melt was often higher than that of the flux on top, because of the poor heat conduction in a heavy layer of molten flux. In the results of experiments where volatile fluxes were used the temperatures given are those of the upper part of the pool of coalesced metal, taken just below the overlying layer of dross, at the time the crucible was taken from the furnace; when other fluxes were used the temperatures given are those of the layer of flux.

### MELTING WITH VOLATILE FLUXES.

The results of a series of experiments in which aluminum borings were melted with various volatile fluxes were as follows:

#### *Results of melting borings with volatile fluxes.*

##### NO FLUX USED.

Run No.	Chips from barrel—	Observ-er.	Weight of chips used.	Flux used, percentage of weight of chips.	Approximate final temperature.	Yield of metal. <sup>a</sup>		
						Primary yield, based on weight of chips.	Total yield, based on weight of chips.	Total yield, on basis of metal content of chips.
1.....	A	G	Kilogram. 2	.....	° C. 910	Per cent. 41	Per cent. 49	Per cent. 54.5

##### NH<sub>4</sub>Cl FLUX USED.

2.....	A	G	2	2	800	62.5	70	b 77
3.....	A	G	2	2	725	45	60.5	67
4.....	A	G	2	5	800	59	65.5	72.5
5.....	A	G	1	2	850	54	60.5	67
6.....	A	G	1	10	850	61	64.5	72
7.....	A	J	1	3	.....	46	49.5	b 55
Average.....	.....	.....	.....	.....	.....	.....	62.5	69.5

##### ANHYDROUS ZnCl<sub>2</sub> FLUX USED.

8.....	A	G	2	2	875	61	66.5	b 74
9.....	A	G	2	2	760	46	62	69
10.....	A	J	1	3	800	15.5	48.5	b 54
Average.....	.....	.....	.....	.....	.....	.....	59	66

##### ANHYDROUS AlCl<sub>3</sub> FLUX USED.

11.....	A	G	2	2	850	53.5	65	72
---------	---	---	---	---	-----	------	----	----

<sup>a</sup> The yields are figured only to the nearest one-half per cent.

<sup>b</sup> The borings were not of uniform size in different parts of the barrel, and were difficult to mix thoroughly. Runs 2 and 8 were companion runs on a part that consisted mostly of borings larger than the average, and runs 7 and 10 were companion runs on some borings smaller than the average. The runs were not made in the order given in the table.

In all of these runs borings were packed tightly into the previously hot crucible, poked down as fast as possible, and more borings added with frequent poking and stirring till the whole charge was in. The charge was then stirred from time to time until the end of the melt, when the caked dross that had collected on the sides and bottom of the crucible below the surface of the pool of metal was scraped up, the fluxes were stirred vigorously into the dross, the crucible taken from the furnace, the dross skimmed into water, and the metal poured. If the prices of the three volatile fluxes used be taken into consideration there is not much choice among them on the basis of these results. On a larger scale, the zinc chloride would probably be preferable, as it does not decompose at so low a temperature as the other fluxes, and hence can be stirred into a large mass of dross without much of it volatilizing before it can be stirred in.

#### MELTING IN ATMOSPHERE OF CHLORINE-CONTAINING GASES.

As the effect of  $\text{NH}_4\text{Cl}$ ,  $\text{ZnCl}_2$ , and  $\text{AlCl}_3$  might be due to the chlorine given off in their decomposition, melting chips in an atmosphere of chlorine or of dry hydrochloric acid gas was tried. The results were as follows:

##### *Results of melting in presence of chlorine and hydrochloric acid gases.*

	Run No. 12.	Run No. 13.
Barrel from which chips were taken.....	A	A
Observer.....	G	G
Weight of chips used, kilograms.....	2	2
Gas used.....	$\text{Cl}_2$	$\text{HCl}$
Approximate final temperature, $^{\circ}\text{C}$ .....	810	825
Yield of metal:		
Primary yield, percentage of weight of chips.....	41	32.5
Total yield, percentage of weight of chips.....	56	56
Total yield, percentage of metal content of charge...	62	62

In run 12,  $\text{Cl}_2$  gas from a cylinder was run into the top of the crucible, which was open, and the charge was poked and stirred as in runs 1 to 11. In run 13, dry  $\text{HCl}$  gas was led in through a hole in the crucible cover and the crucible was opened only to poke the charge down and add more borings. Some  $\text{AlCl}_3$  was formed in this run.

#### MELTING WITH CONTINUOUS PUDDLING.

After it was found, late in the investigation, what great stress the refining company that used the iron-pot method with continuous stirring or puddling laid on keeping the temperature down and on the necessity of continuous stirring, that method was tried on

chips from barrel B, as the contents of barrel A had been used up. The results were as follows:

*Results of melting with continuous puddling.*

	Run No. 14.	Run No. 15.
Barrel from which chips were taken.....	B.	B.
Observer.....	G., J.	J.
Weight of chips used, kilograms.....	1.5	1.5
Flux used.....	ZnCl <sub>2</sub> .	ZnCl <sub>2</sub> .
Quantity of flux used, percentage of weight of chips...	2	2
Approximate final temperature, °C.....	800	800
Yield of metal:		
Primary yield, percentage of weight of chips.....	63.5	54
Total yield, percentage of weight of chips.....	66	60
Total yield, percentage of metal content of charge.	78	71

The temperature was not allowed to rise above 700° C. during the addition of borings. As the furnace was not of the iron-pot type, but a crucible was used setting loosely in the furnace, it was difficult for one observer to stir the pasty mass properly and at the same time hold the crucible in place. The gases of combustion also passed over the crucible, which is not the case in the iron-pot furnace, and hence, as the furnace cover had to be left off to permit stirring and holding the crucible, it was more difficult to prevent oxidation than in the regular iron pot. Run 14 was more thoroughly stirred than run 15.

RETORTING.

In one run the conditions of retorting were approximated by packing 1 kilogram of borings into a hot crucible large enough to contain all the borings at the start. The crucible was then tightly covered and heated without any stirring or poking till the end of the melt, when sal ammoniac was added and the mass stirred and skimmed. The results were as follows:

*Results of retorting aluminum chips (run No. 16).*

Sample used, chips from barrel B.
Observer, G.
Weight of chips used, 1 kilogram.
Flux used, ZnCl <sub>2</sub> .
Quantity of flux used, on basis of weight of chips, 2 per cent.
Approximate final temperature, 900° C.
Yield of metal:
Primary yield, on basis of weight of chips, 58 per cent.
Total yield, on basis of weight of chips, 64 per cent.
Total yield, on basis of metal content of charge, 71.5 per cent.

MELTING IN A REDUCING ATMOSPHERE.

In order to try the effect of melting aluminum scrap in a strongly reducing atmosphere some tests (Nos. 17 to 21) were made in an

electric crucible lift-out furnace of the Hoskins type and some (Nos. 22 and 23) in one of the indirect-arc type, in both of which furnaces gas analyses showed the atmosphere to be strongly reducing when the furnace was closed. These tests were made before the others herein recorded, on a sample of borings (sample C) that were not so dirty as those from barrels A and B. Sample C contained more oil than samples A and B, however. It was estimated that sample C contained about 5 per cent oil and not over 5 per cent dirt. The dross in runs 17 to 23 was not skimmed into water, and the buttons held in it were oxidized completely, so that the results of the tests, which give only the primary yield, would have been increased somewhat by the secondary yield had the buttons in the dross been recovered. The results follow:

*Results of melting borings in a reducing atmosphere.*

Run No.	Sample—	Observer.	Weight of chips used.	Flux used.	Amount of flux used, percentage of weight of chips.	Approximate final temperature.	Furnace used.	Primary yield, percentage of weight of chips.
			<i>Pounds.</i>			<i>° C.</i>		
17.....	C.	G. and L.	12.5	NH <sub>4</sub> Cl..	2	800	Crucible...	40
18.....	C.	do.	14	do.	2	760	do.	55
19.....	C.	do.	29	do.	2	do.	do.	65.5
20.....	C.	do.	24.5	do.	2	do.	do.	63
21.....	C.	do.	28	do.	2	700	do.	52
22.....	C.	do.	40	do.	2	do.	Arc	30
23.....	C.	do.	30	do.	2	do.	do.	50

The melts in the crucible furnace were poked and stirred quite a good deal while the oil contained in the borings was smoking off, as the furnace cover could be opened without danger of getting an oxidizing atmosphere, but thereafter no stirring was done till the final addition of NH<sub>4</sub>Cl just before taking out the crucible. In the runs in the arc furnace no stirring was done, and the furnace door was opened as little as possible in charging. In the arc furnace the local overheating of the borings as soon as charged seemed to cause enough oxidation, through the air held by the borings, to prevent coalescence, so that a large proportion of dross was formed, which entangled much metal.

HEATING IN ABSENCE OF AIR.

In melting borings from sample A in the gas furnace 1 kilogram of borings was rammed down into the bottom of a hot crucible, and 1 kilogram of molten metal at 800° C., from another crucible, was poured on top of them so as to cover them completely and then the whole was heated. The vapor from the oil in the borings bubbled up through the molten metal that remained on top. No stirring was



done till a thermocouple plunged to the bottom of the crucible showed a temperature of  $800^{\circ}\text{C}$ ., when the whole was stirred and much dross that had collected on the sides and bottom of the crucible was brought to the top.  $\text{NH}_4\text{Cl}$  was then stirred in and the crucible skimmed into water. The results of the test were as follows:

*Results of melting borings in absence of air (run No. 24).*

Sample used, chips from barrel A.

Observer, G.

Weight of chips used, 1 kilogram.

Flux used,  $\text{NH}_4\text{Cl}$ .

Quantity of flux used, 2 per cent.

Approximate final temperature,  $800^{\circ}\text{C}$ .

Yield of metal:

Primary yield, on basis of weight of chips, 50 per cent.

Total yield, on basis of weight of chips, 61 per cent.

Total yield, on basis of metal content of charge, 68 per cent.

USING A HEEL OF MOLTEN METAL.

Another run was started with a "heel" of 1 kilogram of molten metal in the bottom of the crucible. One kilogram of borings was added a little at a time, an attempt being made to poke them down continually below the surface of the fluid metal—that is, to follow the usual directions for using a heel of metal. This procedure differs from the puddling process in that there is fluid metal in the bottom of the pot, whereas in puddling the whole mass is kept pasty during the addition of borings.

The chips floated on the metal and obstinately refused to stay submerged. The temperature was kept rather low throughout and was, as it proved, not raised sufficiently at the end to give a good separation of metal from dross. The results were as follows:

*Results of melting chips with heel of molten metal (run No. 25).*

Sample used, chips from barrel A.

Observer, G.

Weight of chips used, 1 kilogram.

Flux used,  $\text{NH}_4\text{Cl}$ .

Quantity of flux used, on basis of weight of chips, 2 per cent.

Approximate final temperature,  $700^{\circ}\text{C}$ .

Yield of metal:

Primary yield, on basis of weight of chips, 39 per cent.

Total yield, on basis of weight of chips, 59.5 per cent.

Total yield, on basis of metal content of charge, 66 per cent.

Other tests on this method and some of the other methods described will be given under the results of sizing and briquetting the chips.

## CONCLUSIONS AS TO RESULTS OF TESTS.

In all of the tests described in the preceding pages the fluxes used were volatile and did not form any slag that would tend to protect the metal from oxidation; the dross skimmed off was voluminous, loose, and contained many tiny metallic globules. If any considerable mass of this dross is left in the air it becomes white hot, and then cakes, the loose particles of dirt and oxide being compactly cemented together by oxide formed from the oxidation of the metal in the buttons. No metallic buttons will be found in the mass after cooling in air unless the globule was originally at least the size of a marble and the coating of oxide about it happens to have been so impervious to air that the inside of the globule is protected. Each tiny globule becomes oxidized throughout. The cooled dross is gray in the mass from the presence of dirt, but many tiny globular white specks of aluminum oxide, pseudomorphic after the metal, show where tiny globules of molten metal were contained in the dross when skimmed off.

By showering the hot dross with its entangled globules of metal through the air slowly one can produce an almost continuous sheet of flame. It is impossible to skim the dross into water quickly enough to prevent the burning up of the smallest globules, as exposure for a fraction of a second to the air is sufficient for their oxidation and nitridation. When any considerable amount of dross is skimmed into water there is a very strong odor of ammonia mixed with hydrocarbons. The gas bubbles that arise from the hot dross as it touches the water take fire spontaneously. It appears that there is a good deal of nitrogen and a little carbon chemically combined with the dross.

Although in the last analysis the loss of metal in these methods of melting borings is due to oxidation and nitridation, yet very little of this loss takes place within the furnace in commercial practice in melting in a pit furnace or in an iron-pot furnace, if the pot is covered at the end, or in the gas furnace used in the tests. The dross does not begin to show the thermit-like glowing until it comes in contact with the air. If each and every one of the tiny globules could be made to coalesce in the crucible and to free itself entirely from the dirt and from the small quantity of oxide that actually forms in the furnace, the loss would be very small. The root of the trouble lies in the failure of the tiny, light globules to coalesce.

With this premise it becomes plain why the presence of fine dirt makes the recoveries low, and why, with a given amount of dirt, the smaller the chips are the lower is the recovery. The greater the amount of dirt and the more finely divided the dirt is, the greater is its power to form the honeycomblike coating which envelops the

globules and prevents their uniting. The tinier the chip the greater is the surface of the globule in respect to its weight. The less the weight of the globule the less is the force tending to tear through the enveloping film.

When the borings are even slightly oily, any dust in the borings sticks tightly to them and a surface film of dust tends to be formed before the chip is melted that under the effect of surface tension readily forms a bag about the tiny drop into which the chip changes on fusion.

The small size of some of the chips can hardly be prevented, as they come from the light finishing cut, but the content of dirt can be greatly reduced if care is taken in the collection of the chips.

The tendency toward oxidation of fine chips can be more easily and more readily checked in melting than their reluctance to coalesce can be overcome. It does not seem that, without the use of some slaglike flux to act as a collector of the tiny buttons, such methods as retorting, vacuum-furnace melting, etc., which aim at preventing oxidation, but do not provide for stirring, or other means of promoting coalescence, are likely to offer any advantage in the melting of dirty chips, at least, commensurate with their cost and the difficulties of their operation.

It would therefore appear that if some menstruum or slag were present through which the globules would drop, they might coalesce better, as the dirt would tend to be soaked up by the slag. There would no longer be a honeycomblike coating of solid dirt enveloping the globules, but the globules and the dirt would both be dispersed in a liquid slag. The slag, if lighter than the metal, would also prevent access of air. Hence it seems possible that a molten cover, not necessarily acting as a solvent of dirt and oxide, but merely soaking them up and taking them away from the globules might both promote coalescence and prevent oxidation. It is plain that the most efficient menstruum will be one which is fluid below the melting point of the metal, and which has not only a lower specific gravity than the metal, but also has a low viscosity, so as to allow the globules to sink readily through it.

#### MELTING WITH MOLTEN COVER.

Suitable molten covers combining these properties with cheapness and with chemical inertness to the metal are, however, not easy to find. Alkali and alkaline-earth chlorides, or mixtures thereof, form the basis of most of the molten covers suggested.

## TESTS WITH A COMMERCIAL FLUX.

One such flux is on the market; it has been mentioned, with the makers' directions for using it in melting ingot, on page 17. Although some aluminum founders who have tried this flux reported unfavorably on its use for melting ingot, one even denouncing it as a fraud, some of it was obtained and tried on borings. The flux is a lavender-colored granular material, is soluble in water, tastes like potassium chloride, and melts at about 620° C. The lavender color is due to some organic dye, as it may be extracted by absolute alcohol, in which the material itself is not soluble. The color burns off on ignition, before the melting point is reached. It is perhaps added for psychological effect on the purchaser. Qualitative analysis shows the presence of K, Na, Cl, SO<sub>4</sub>, and a trace of Mg. Quantitative analysis was not made, but it appears that a molecular mixture of KCl and NaCl, plus a little K<sub>2</sub>SO<sub>4</sub> would give a mixture of about the same melting point and properties.

In all the following tests on molten covers and other fluxes, the powdered flux was mixed with the borings before charging, and the dross or slag, except where so stated in the tables, was not skimmed directly into water, on account of the explosions that occur in such a case. The dross or slag mixed with dross was skimmed off, allowed to cool somewhat, and thrown into water while still hot in order to disintegrate it. It was then crushed, washed on a 20-mesh sieve, and hand picked. If only a few large clean buttons were obtained, their weight was added to the weight of ingot to give the total yield. If the buttons were small and adhered to particles of slag or dross, they were remelted with the same proportion of the same kind of flux as was used in the original melt and the weight of secondary ingot thus obtained was added to that of the original ingot to give the total yield.

The results of the test with the commercial flux mentioned were as follows:

*Results of tests with commercial flux.*

Run No.	Chips from barrel—	Observer.	Charge used.	Flux used, percentage of weight of chips.	Approximate final temperature.	Yield of metal.		
						Primary yield, on basis of weight of chips.	Total yield, on basis of weight of chips.	Total yield, on basis of metal content of charge.
			Kg.		° C.	Per cent.	Per cent.	Per cent.
26.....	A	G	2	5	850	45.0	58.0	64.0
27.....	A	G	2	25	760	48.5	62.0	69.0
28.....	B	J	1	25	900	55.0	62.5	73.5



## TESTS WITH DEHYDRATED CARNALLITE.

Dehydrated carnallite, or a mixture in molecular proportions of KCl and  $\text{MgCl}_2$ , has a low melting point and has been suggested as a flux. Tests with this flux gave the following results:

*Results of melting borings with flux of dehydrated carnallite.*

	Run No. 29.	Run No. 30.
Barrel from which chips were taken.....	A	A
Observer.....	G	G
Weight of chips used, kilograms.....	2	2
Percentage of flux used, on basis of weight of chips. . . .	5	25
Approximate final temperature, °C.....	725	800
Yield of metal:		
Primary yield, percentage of weight of chips.....	47	48
Total yield, percentage of weight of chips.....	60.5	59
Total yield, percentage of metal content of charge..	67	65.5

## DISCUSSION OF RESULTS WITH THE TWO FLUXES.

Neither flux, even when 25 per cent was used, was in sufficient quantity to soak up all the dirt and dross and still become truly fluid. Neither flux rose entirely to the top, but much stuck to the sides and bottom of the crucible.

## SPRINKLING WITH ZINC-CHLORIDE SOLUTION AND DRY SALT.

As the use of 2 per cent of a strong solution of  $\text{ZnCl}_2$  and 2 per cent dry NaCl sprinkled over the borings had been suggested (see p. 30), this was tried. The results of the test were as follows:

*Results of melting borings sprinkled with  $\text{ZnCl}_2$  solution and dry NaCl (run No. 31).*

Barrel from which chips were taken, A.

Observer, J.

Weight of chips used, 1.5 kilograms.

Quantity of flux used, 2 per cent of  $\text{ZnCl}_2$  solution, 2 per cent of dry NaCl.

Approximate final temperature, 1000° C.

Yield of metal:

    Primary yield, on basis of weight of chips, 45 per cent.

    Total yield, on basis of weight of chips, 58 per cent.

    Total yield, on basis of metal content of chips, 64 per cent.

The slag was very sticky, almost solid with the dross and dirt soaked up by it, so that it was necessary to heat it very hot to get any separation of metal from it. The slag and dross showed a thermit-like reaction when exposed to air.

## TESTS WITH COMMON SALT.

As common salt has been used in commercial practice and is the cheapest alkali chloride, a series of tests was made with varying

proportions of an "agricultural grade" of salt, costing \$2 per ton at the salt works, as flux. The temperatures given are the temperatures in the slag layer at the end of the heat, not in the metal. The results of the tests follow:

*Results of melting borings with common salt.*

Run No.	Chips from barrel—	Observer.	Charge used.	Flux used, percent- age of weight of chips.	Approximate final temperature.	Yield of metal.		
						Primary yield, on basis of weight of chips.	Total yield, on basis of weight of chips.	Total yield, on basis of metal content of charge.
			<i>Kg.</i>		<i>° C.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
32.....	A	J	1	15	900	45.5	62	69
33.....	A	J	2	20	900	66	69.5	77
34.....	A	J	1	25	900	48.5	61	68
35.....	A	G	1.5	50	900	66	76.5	85
36.....	A	J	1	50	900	59	71	79
37.....	A	J	1	50	900	54.5	61.5	68.5
38.....	A	J	1	50	900	59	65	72
39.....	A	J	1	100	900	47	66	73

With as little as 15 per cent salt the dross was not all soaked up and the mass was dry in places and not wholly pasty and the thermit-like reaction was not wholly stopped. The separation was not good. With 100 per cent of flux the volume of slag was too great to give good separation; 20 to 50 per cent of salt gave pasty slags that were easily skimmed off, gave fair separation, and prevented the thermit-like reaction, although there was not perfect coalescence, as vast numbers of tiny buttons were still held in the slag, passed the 20-mesh sieve, and were lost.

TESTS BY A FOUNDRY WITH SALT AND CALCIUM CHLORIDE FLUX.

Some tests by a foundry, reported to the writers, on NaCl, CaCl<sub>2</sub>, and a mixture of the two, as flux, are given below.

SERIES 1, MELTING EMERY GRINDINGS.

Some experimental melts were made in a small crucible, oil-fired furnace; the material used was sawdust from band saws used in cutting off gates, etc., plus emery grindings and emery dust from the grinding wheels. The probable metallic content was between 60 and 70 per cent, the rest being emery dust, floor sweepings, and paraffin used as a lubricant in sawing and grinding. The material would nearly all pass a 20-mesh sieve. The results were as follows:

*Charge used.*—1½ kilograms.

*Run A.*—No flux was used. The total recovery of metal was 19 per cent on original weight, all as ingot, no metal being recovered from the dross, as this burnt up on removal.

*Run B.*—In this run 75 per cent NaCl was used as flux. The total recovery of metal was 47 per cent. Much of this metal was from buttons in the dross, which was not fluid, but was very pasty, almost dry.

*Run C.*—In this run 30 per cent commercial fused  $\text{CaCl}_2$  flux was used. The flux melted and dust was poked constantly into it. The recovery was 43 per cent in good ingot.

*Run D.*—In this run 75 per cent flux, which was two-thirds commercial fused  $\text{CaCl}_2$ , one-third  $\text{NaCl}$ , was used. The total recovery was 38 per cent, mostly in ingot; some metal was recovered as large buttons in the flux cover.

*Run E.*—This run was made on 60 per cent of old flux, from run D, containing a few aluminum shots from that charge which were not picked out. The recovery was 44 per cent, but most of the metal was buttons from the flux, which was not sufficiently fluid on re-using.

SERIES 2, MELTING DIRTY BORINGS WITH SAL AMMONIAC AND SODIUM CHLORIDE —  
CALCIUM CHLORIDE FLUXES.

A comparison of sal ammoniac and the  $\text{NaCl}$ — $\text{CaCl}_2$  mixture as fluxes was made. Exceptionally small, thin, oily, and dirty borings, of unknown metallic content, of very poor quality, were melted in 100-pound charges in an oil-fired iron pot.

*Run A.*—The borings were constantly poked, and 1 per cent sal ammoniac was used at the end of the melt. The recovery was  $12\frac{1}{2}$  per cent.

*Run B.*—The borings were mixed with 20 per cent of the  $\text{CaCl}_2$ — $\text{NaCl}$  flux, and were poked occasionally. The recovery was 42 per cent.

SERIES 3, TESTS ON CLEANER BORINGS WITH SAL AMMONIAC AND SODIUM CHLORIDE —  
CALCIUM CHLORIDE FLUXES.

A similar comparison of the two fluxes was made on commercial borings, of much better quality than in series 2, but containing some oil and dirt. The borings were melted in crucibles in coke fires.

Charges aggregating 316 pounds were melted, with 1 per cent of sal ammoniac as flux. The recovery was 144 pounds, or 44 per cent. On other charges aggregating 346 pounds, with 26 pounds of fused commercial  $\text{CaCl}_2$  and 16 pounds  $\text{NaCl}$ , the recovery was 194 pounds, or 56 per cent.

The flux did not meet with favor by those running the remelting work, largely because of its hygroscopic nature and the consequent difficulty of avoiding dangerous explosions when it is used.

TEST WITH MIXTURE OF CALCIUM CHLORIDE AND SODIUM CHLORIDE.

Chips from lot A were melted with 50 per cent of a flux consisting of 90 per cent commercial fused  $\text{CaCl}_2$ , which contained quite a proportion of  $\text{NaCl}$ , and 10 per cent added  $\text{NaCl}$ , the mixture becoming fluid at  $600^\circ \text{C}$ . The results of the tests were as follows:

*Results of melting borings with flux of  $\text{CaCl}_2$  and  $\text{NaCl}$  (run No. 40).*

Barrel from which chips were taken, A.

Observer, G.

Weight of chips used, 1.5 kilograms.

Quantity of flux used, 50 per cent (90 parts  $\text{CaCl}_2$ , 10 parts  $\text{NaCl}$ ).

Approximate final temperature,  $900^\circ \text{C}$ .

Yield of metal:

Primary yield on basis of weight of chips, 61 per cent.

Total yield on basis of weight of chips, 72 per cent.

Total yield on basis of metal content of chips, 80 per cent.

The flux, though fluid at  $600^\circ \text{C}$ ., was only just thin enough to skim off properly at  $900^\circ \text{C}$ . because of the dirt soaked up by it.

## FLUXES CONTAINING FLUORIDES.

In order to get a flux that will not only act as a molten cover to protect the metal from oxidation and a collector to soak up the dirt and permit the globules to coalesce, but will also have a true fluxing or dissolving action both on the dirt and the aluminum oxide, the use of fluorides, as suggested by Schoop,<sup>a</sup> was tried.

## TESTS WITH VARIOUS FLUORIDE FLUXES.

Various fluoride fluxes were first used, to find out whether better recoveries were obtainable than with the molten covers. Flux No. 1 was as follows: Fluorspar ( $\text{CaF}_2$ ) 16 parts, cryolite ( $\text{AlF}_3\text{--NaF}$ ) 28 parts, aluminum fluoride ( $\text{AlF}_3$ ) 56 parts, this being, according to Wright,<sup>b</sup> the composition of the electrolyte used in the production of aluminum. The constituents were fused together, cooled, and the mass pulverized.

Plato<sup>c</sup> shows that a mixture of about 86 parts  $\text{CaCl}_2$  and 14 parts  $\text{CaF}_2$  melts at about  $650^\circ\text{C}$ .

Such a mixture (flux No. 2) was made, fused, and pulverized. It was fairly fluid at about  $700^\circ\text{C}$ , and the  $\text{CaF}_2$  seemed to be all dissolved.

Gleason<sup>d</sup> suggests a flux of fluorspar and anhydrous boric acid, in the proportions of three parts of fluorspar to one of boric acid, for another purpose, that of introducing boron into copper by treating such a flux with aluminum and adding it to copper. It was thought that this mixture might give a fluoride flux for the purpose in question, but it was not sufficiently fluid, even at  $1100^\circ\text{C}$ ., so it was not used.

Jones<sup>e</sup> suggests a flux consisting of equal parts  $\text{LiCl}_2$ ,  $\text{KCl}$ , and  $\text{NaF}$ , for use in welding aluminum. Such a flux (No. 3) was made except that  $\text{KF}$  was used instead of  $\text{NaF}$ , no  $\text{NaF}$  being at hand, then fused and pulverized. This flux was extremely fluid at a low red heat.

A flux consisting of 10 parts  $\text{Na}_2\text{CO}_3$ , 2 parts  $\text{K}_2\text{CO}_3$ , 2 parts cryolite, and 1 part borax has been suggested<sup>f</sup> to be used in the proportion of  $1\frac{1}{2}$  pounds per hundredweight of chips. The content of fluorine is so small that this can scarcely be classed as a fluoride flux. Such a flux (No. 4) was made, fused, and pulverized.

Powdered cryolite flux (No. 5) was also tried.

<sup>a</sup> Schoop, M. V., U. S. Patent 943,164, Dec. 19, 1909.

<sup>b</sup> Wright, J., *Electric furnaces and their industrial applications*, 1908, pp. 186-187.

<sup>c</sup> Plato, W., *Erstarrungs erscheinungen an anorganischen Salzen und Salzgemischen*: Ztschr. phys. Chem. Bd. 58, 1907, p. 362.

<sup>d</sup> Gleason, E. D., U. S. Patent 1,076,973.

<sup>e</sup> Jones, J. L., *Shop problems—fluxing*: Metal Ind., vol. 11, 1913, p. 523.

<sup>f</sup> Anon, *Melting aluminum chips, Castings*, vol. 38, 1910, p. 78.



The results of the experiments with the five fluxes named follow:

*Results of melting borings with fluxes containing fluorides.*

Run No.	Chips from barrel—	Observer.	Charge.	Flux used.	Amount of flux, on basis of weight of chips.	Approximate final temperature.	Yield of metal.		
							Primary yield, based on weight of chips.	Total yield, based on weight of chips.	Total yield, based on metal content of chips.
			Kg.		Per cent.	° C.	Per cent.	Per cent.	Per cent.
41.....	A	G	1	No. 1	25	900	71	72	80
42.....	A	G	1	No. 2	25	900	64	71	79
43.....	A	G	1	No. 3	25	900	65	69.5	78
44.....	A	G	1	No. 4	1.5	900	54	65	72
45.....	A	G	1	No. 4	20	900	50	62	70
46.....	A	J	1	No. 5	15	900	68.5	71	79
47.....	A	G	1	No. 5	25	900	70	71.5	79.5

In runs Nos. 44 and 45, with the soda ash, pearl ash, borax, cryolite flux, which is low in fluorine, the dross showed a thermit-like reaction, whereas the other fluxes gave a pasty mixture of dross and flux which cooled without such reaction. The four fluxes, Nos. 1, 2, 3, and 5, gave consistent results, better than the average results from any other of the fluxes used thus far.

The foundry reporting the results on tests with molten covers also stated that a small laboratory scale test in a crucible in a gas-fired furnace, in which millings free from oil or dirt, being cut without lubricant as a sample for analyses, and rather larger than commercial borings and free from very fine material, with a large proportion of powdered cryolite as flux, had given a recovery of 94½ per cent, although some of the metal was in the form of buttons, as the temperature was not high enough to make the cryolite sufficiently fluid.

TESTS WITH SALT AND FLUORSPAR FLUX.

As the writers learned at this point in the work that a flux of salt and fluorspar was being used commercially, especial attention was directed to that flux, on account of its cheapness, although its melting point is considerably above that of aluminum or its common alloys. As definite information on the proportions of the two ingredients used was not available, experiments were made on various mixtures of the commercial salt and fluorspar that were on hand. The freezing points of mixtures ranging from 90 parts NaCl and 10 parts CaF<sub>2</sub> (about 790° C.) to 75 parts NaCl and 25 parts CaF<sub>2</sub> (about 800° C.) were studied. The mixture of 85 parts NaCl and 15 parts CaF<sub>2</sub>, which froze at about 785° C., had the lowest melting point and did not give so viscous a melt as those higher in CaF<sub>2</sub>, being quite fluid at 800° C., was chosen as the most suitable composition. One mixture of 85 parts commercial CaCl<sub>2</sub> (containing NaCl) and 15 parts CaF<sub>2</sub>,

froze at about 660° C. On substituting 5 and 10 per cent of NaCl for similar percentages of CaCl<sub>2</sub>, mixtures freezing at about 675° C. and 680° C. were obtained, but all of the CaCl<sub>2</sub> mixtures seemed very viscous at temperatures well above their fusion points and they did not seem to be better fitted for the purpose than the salt and fluorspar mixture, notwithstanding their lower freezing points, and they had the disadvantage of being hygroscopic.

In these runs the flux was intimately mixed with the chips before charging. The borings were rammed down as soon as they got soft and pasty so as to make room for more borings. After the whole charge was in, the crucible was not disturbed or stirred until the temperature of the flux on top of the metal was 900° C., at which temperature it is fluid enough, unless there is too much dirt present in proportion to the amount of flux used, when higher temperatures are needed to make the mass really fluid. Then the crucible was taken from the furnace, stirred gently—too violent stirring will carry globules of metal up into the slag from the metal below—skimmed, and poured. The buttons were recovered from the slag as in the previous experiments.

*Results of melting borings with flux consisting of 85 per cent NaCl and 15 per cent CaF<sub>2</sub>.*

Run No.	Chips from barrel—	Observer.	Charge used.	Quantity of flux, based on weight of chips.	Approximate final temperature.	Yield of metal.		
						Primary yield, based on weight of chips.	Total yield, based on weight of chips.	Total yield, based on metal content of charge.
			Kg.	Per cent.	°C.	Per cent.	Per cent.	Per cent.
48.....	A	J	1	15	900	57.5	64	71
49.....	A	J	2	25	900	68.5	70	78
50.....	A	J	2	25	900	74	75	83.5
51.....	A	J	1	25	900	69	72	80
52.....	A	G	1.5	50	900	70	76	84.5
53.....	A	J	1	50	900	63	68	75.5
54.....	A	J	1	100	900	53	63.5	70.5
55.....	A	J	1.5	20	900	61	65	72

In run 55 the borings and flux were puddled, as in melting with ZnCl<sub>2</sub> in the iron-pot method, and kept pasty by continued stirring till all the charge was in, then at the end the temperature was raised. Apparently nothing is to be gained by so doing, as compared with merely ramming the borings down to make room for the whole charge, and not stirring till the end.

The behavior with only 15 per cent flux or with as much as 100 per cent was analagous to that with similar amounts of salt alone. About 25 to 50 per cent of this flux appears advisable for borings carrying the amount of dirt contained in the lot melted.

## SUMMARY OF RESULTS WITH VARIOUS METHODS OF MELTING.

A summary of the results of some of the experimental runs on this consignment of dirty chips, by the chief methods that have been suggested, is shown below:

*Results of melting borings by various methods.*

No. of run.	Flux or method used.	Number of tests.	Average total recovery, on basis of metal content of charge.	Maximum.
			<i>Per cent.</i>	<i>Per cent.</i>
1	No flux used .....	1	54.5	-----
2 to 7	NH <sub>4</sub> Cl flux used .....	6	69.5	77
8 to 10	ZnCl <sub>2</sub> flux used .....	3	66	74
11	AlCl <sub>3</sub> flux used .....	1	72	-----
14, 15	Puddling, with NH <sub>4</sub> Cl added at end of melt .....	2	74.5	78
16	Retorting, with NH <sub>4</sub> Cl added at end of melt .....	1	71.5	-----
26 to 30	Various molten covers, other than NaCl used .....	5	66	73.5
32 to 39	NaCl in proportions of 15 to 100 per cent. ....	8	74	85
33 to 38	Common salt in proportions of 20 to 50 per cent used .....	6	75	85
40	NaCl-CaCl <sub>2</sub> flux used .....	1	80	-----
41 to 47	Various fluoride fluxes other than NaCl-CaF <sub>2</sub> used .....	7	79.5	80
48 to 55	NaCl-CaF <sub>2</sub> flux used .....	8	75	84.5
49 to 53, 55	NaCl-CaF <sub>2</sub> flux in proportions of 20 to 50 per cent used .....	5	80	-----

The best results appear to come from the three following methods: Using NaCl, NaCl plus CaCl<sub>2</sub> in rather large proportion, or from the fluoride-containing fluxes in rather large proportion. Of the fluoride-containing fluxes the cheapest, the NaCl-CaF<sub>2</sub> flux, is apparently as good as those more expensive and seems to give, on an average, sufficiently better results than NaCl without CaF<sub>2</sub>, to justify the use of the CaF<sub>2</sub>.

In the type of furnace used for the tests, which was not properly adapted to puddling, the use of salt and fluorspar without puddling gave better results than puddling and with much less labor.

This flux was then tested out experimentally under other conditions on briquetted material in a small reverberatory furnace and in an electric furnace.

## EXPERIMENTS WITH BRIQUETTED CHIPS.

As is brought out in page 19, compacting the borings into briquets under heavy pressure has been suggested on the theory that such briquets could be more readily plunged under a heel of molten metal than the loose borings, and that the briquetted metal would act much like ingot metal.

However, one refining plant, melting in coke-fired pit furnaces and using sal ammoniac as flux, had some borings briquetted. The briquets were very solid and compact, and could be thrown about without breaking. No quantitative data are available on the results obtained with them; but it was reported that they disintegrated on



melting, and, save for greater ease in charging and possibly greater speed in melting, their use had no advantage over charging loose borings, as the loss was about the same.

A firm selling briquetting machines sent the writers two briquets, stating that they were made from a class of shop fines of No. 12 aluminum containing, unavoidably, a considerable amount of dirt, and that "this class of briquets refused to melt without unreasonable loss." These briquets, which were approximately disks or flat cylinders 3 inches in diameter, 1 inch thick at the edge and  $1\frac{1}{2}$  inches thick at the center, were very solid and had apparently been made under extremely high pressure. The briquets were run down with results as follows:

#### RESULTS OF MELTING BRIQUET NO. 1.

In run No. 56 a heel of metal, consisting of 693 grams of clean No. 12 ingot, was melted in a No. 5 crucible in a gas furnace and raised to a temperature of  $800^{\circ}\text{C}$ ., when one of the sample briquets, weighing 342 grams, was put in. It sank in the molten metal, a smoky flame, evidently caused by the cutting oil used in machining the castings from which the chips came, being given off. The briquet was poked from time to time with a graphite rod. It disintegrated to some extent and much light dross rose to the surface. Seven minutes after charging the bath was again at a temperature of  $800^{\circ}\text{C}$ ., and 7 grams of anhydrous zinc chloride was added and stirred into the dross. The dross showed a tendency to glow and heat up, similar to the thermit reaction, indicating that there were still very tiny globules of metal in the dross. The dross was skimmed off into water and the metal poured into ingot. Although the metal had been well stirred, much dross stuck to the bottom and sides of the crucible. After pouring the metal this dross was also skimmed into water. The dross was then washed on a 14-mesh sieve and all buttons over 14 mesh were collected.

In all, 920 grams of ingot were poured and 7 grams of buttons were collected. Deducting the 693 grams of heel metal, there was recovered from the original 342-gram briquet 227 grams of ingot and 7 grams of buttons, or the total recovery was 234 grams. On an assumed basis of 90 per cent metal content and 10 per cent oil and dirt in the briquet, the results are as follows:

#### *Results of melting briquet in heel of molten metal (run No. 56).*

	Per cent.
Primary yield, on basis of weight of briquet.....	66.5
Secondary yield, on basis of weight of briquet.....	2.0
Total recovery, on basis of weight of briquet.....	68.5
Total recovery, on assumed basis of 90 per cent metal content.....	76.2



This recovery is not much above the results from loose borings of apparently the same quality as those in the briquet when melted by this method.

#### RESULTS OF MELTING BRIQUET NO. 2.

Briquet No. 2 was melted with a flux consisting of 85 per cent common salt and 15 per cent fluorspar. As the flux could not be intimately mixed with the briquet, as can be done with loose borings, it was decided to cover the briquet with the flux. In this run (No. 57) it was impossible to cover the briquet in a crucible with so little as 20 to 30 per cent of its weight of this flux, the amount required for loose borings. Accordingly, 200 grams of flux was melted in a No. 5 crucible, and when it had reached a temperature of 800° C. the briquet, weighing 345 grams, was charged. As the flux did not quite cover the briquet, 100 grams more flux was added on top. In 12 minutes the briquet and the added flux had melted and the whole was at 875° C. The flux was then poured off, the metal poured into ingot, and the pasty mass of dirt which had soaked up flux and stuck to the sides of the crucible on pouring was scraped out. The flux and scrapings were disintegrated in water, after being permitted to cool somewhat, and washed on a 14-mesh sieve. In all, 265 grams ingot and 15 grams buttons were recovered, a total of 280 grams. Or the recovery was as follows:

#### *Results of submerging briquet in molten flux (run No. 57).*

	Per cent.
Primary yield, on basis of weight of briquet.....	76.8
Secondary yield, on basis of weight of briquet.....	4.2
Total recovery, on basis of weight of briquet.....	81.0
Total recovery, on assumed basis of 90 per cent metal content.....	90.0

#### CONCLUSIONS AS TO TESTS.

This recovery is better than that obtained on the loose dirty borings of lot A, but the briquets, although they contained some oil and dirt, did not appear to contain quite so much as lot A, hence no conclusions can be drawn from this single small-scale test as to any decrease in melting loss due to briquetting.

Briquetting will increase the ease of handling and probably increase the speed of melting, as the solid briquets transmit heat more rapidly than loose material. Whether it would increase the recovery or not, and whether the cost of briquetting, including labor, power, interest, and upkeep on an expensive press, would be compensated for by increased recovery and decreased labor and fuel cost in melting, can only be determined by large-scale tests on loose and briquetted chips from the same consignment.

In order to utilize the salt-fluorspar flux, it might be worth while to mix 20 to 30 per cent of flux, depending on the amount of dirt in the borings, with dirty borings before briquetting, thus making a briquet with the flux just where it is needed.

#### REVERBERATORY MELTING.

One firm is refining aluminum borings in a reverberatory furnace and claims 80 per cent recovery. The details are not available as the refiners claim to have a "secret process." Reverberatory melting is also done to a small extent by the firm using the  $\text{NaCl-CaF}_2$  flux in crucible melting. This flux is also used when they melt in the reverberatory. Reverberatory melting of aluminum borings is considered by many refiners as out of the question, because of loss by oxidation in melting; also most of the firms melting aluminum ingot consider reverberatory or open-flame melting impossible for the same reason. On the other hand ingot may be successfully melted in an open-flame furnace if a strongly reducing flame is used, and the market alloys put out by the manufacturers of aluminum are usually made in large reverberatory or tilting open-flame, natural gas, or oil furnaces.

In order to determine whether the salt-fluorspar flux, which in crucible melting has consistently given better results than any other flux or method used in the experimental tests, was applicable to reverberatory melting, a small make-shift reverberatory or tilting open-flame gas furnace was built, an old carboy rocker being used for the tilting device, and an old sink for the body of the furnace.

The sink was lined with fire brick and a hearth laid within the fire-brick shell and lined with alundum-magnesite mixture. The hearth proper was about 2 inches deep at the lowest point,  $2\frac{1}{2}$  inches wide, 10 inches long on the flat part of the bottom, and sloped up to 5 inches wide by 16 inches long at the top of the hearth. A pouring spout was made at one end and space for a charging hole left above the hearth at the other. An arched fire-brick roof was added, the highest point of which was about 5 inches from the flat bottom of the hearth, or 3 inches above the level of a full molten charge, and 1 inch above the top of the hearth proper at the edges of the hearth on the sides of the furnace. The walls and roof were all 4 inches thick.

City gas, the only kind available, was used to heat the furnace. A home-made burner made from pipe and fittings was placed a few inches in front of the pouring spout. The waste gases passed out the charging hole of the furnace. As the burner was not tightly luted into the mouth of the furnace, but the flame was merely directed into it, air was freely drawn in by injector action. The flame did not heat the furnace very uniformly, the hottest part being about one-third

the way from the front. The rear third of the furnace did not heat up very well, so the furnace was run by tilting it forward a little and utilizing only the front half, the rear being used to preheat the charge, which was poked forward as room was made for it at the front.

It will be seen that it would be difficult to maintain a very strongly reducing flame in such a furnace, so freely open to the air. The temperature varied so throughout the furnace that except on charges melted to a uniform liquid, the temperature of the charge varied with the location in the furnace.

A number of melts of aluminum borings from lot B, containing oil and much fine dirt, with not over 85 per cent metal content, were made. The results of these melts are given below in the order in which they were made:

#### TESTS IN REVERBERATORY FURNACE.

*Run No. 58.*—Four pounds of borings were mixed with 1 pound (85 per cent NaCl, 15 per cent  $\text{CaF}_2$ ) of flux and the whole charged into the furnace, which was fairly hot at the start. The run lasted 22 minutes, no attempt being made to keep the flame strongly reducing. The metal was poured before the flux was really fluid, the temperature in the hottest part of the flux being  $825^\circ\text{C}$ . Apparently the amount of flux was too small, and the charge a little too large, as the direct flame played on the upper part of the charge. Some thermit-like burning of the charge near the front of furnace was noted.

In this run 1.6 pounds of ingot and 2 pounds of dirty buttons over 20-mesh in size, mixed with balled-up masses of dirt and flux, were recovered. The buttons were remelted with 1 pound of flux, yielding 1.05 pounds ingot, and 0.2 pound of large buttons, the total recovery being 2.85 pounds. The primary yield was 40 per cent, the secondary yield 31 per cent, making a total of 71 per cent, or  $83\frac{1}{2}$  per cent on the basis of metal content of charge.

*Run No. 59.*—The whole charge, consisting of 3.2 pounds borings and 1 pound of NaCl- $\text{CaF}_2$  flux mixed with borings, plus 1 pound flux spread on top after charging, was charged at the start in a hot furnace. The furnace was run 25 minutes and the bulk of the metal poured, then run 15 minutes more to melt the flux, but it was pasty at the end of the run. No thermit reaction was noted. The yield was as follows: Primary yield, 2.4 pounds, 75 per cent; 0.05 pound large buttons, 1.5 per cent; total, 2.45 pounds, 76.5 per cent, or 90 per cent on the basis of metal content of charge.

*Run No. 60.*—The whole charge, consisting of 3 pounds of borings with 1 pound of flux spread on top, none mixed in with borings, was charged in the furnace, which was warm, but not hot. The run lasted 25 minutes. At times thermit-like burning of the charge was observed. The dross and flux were very dry, not being fluid or even pasty, though the temperature in the hottest part of the furnace was over  $900^\circ\text{C}$ .

The recovery was as follows: Primary yield, 1.55 pounds, 52 per cent; 0.2 pound of large buttons, or  $6\frac{1}{2}$  per cent; total yield, 1.75 pounds, or  $58\frac{1}{2}$  per cent, or 69 per cent on the basis of metal content of charge.

*Run No. 61.*—Three pounds of borings and 1 pound flux mixed were charged in the hot furnace and  $\frac{1}{2}$  pound of flux spread on top. The run lasted 25 minutes; the flux was pasty at the end. No thermit action was observed. The recovery was as follows: Primary, 2.1 pounds, or 70 per cent; large buttons, 0.05 pound, or  $1\frac{1}{2}$  per cent; total, 2.15 pounds, or  $71\frac{1}{2}$  per cent, or  $84\frac{1}{2}$  per cent on the basis of metal content of charge.

*Run No. 62.*—The whole charge, consisting of 3 pounds of borings and 2 pounds flux, all mixed in, was charged in the hot furnace; ran 20 minutes. The flux was nicely



pasty and a little over 900° C. in the hottest part. No thermit action was noted. The recovery was as follows: Ingot, 2.1 pounds, or 70 per cent; large buttons, 0.05 pound, or 1.5 per cent; total, 2.15 pounds, or 71½ per cent, or 81.5 per cent on the basis of metal content of charge.

*Run No. 63.*—The whole charge, consisting of 5 pounds of borings and 2½ pounds of flux, all mixed in, was charged in the hot furnace; ran 30 minutes. The charge was too large, as the flame played on top of it. Some thermit action was noted. The mixture of flux and dross was too dry.

The recovery was as follows: Ingot, 2.9 pounds, or 58 per cent; large buttons, 0.15 pound, or 3 per cent; total yield, 3.05 pounds, or 61 per cent, or 71.5 per cent on the basis of metal content of charge.

*Run No. 64.*—The whole charge, consisting of 3 pounds of borings and 3 pounds of flux, all mixed in, was charged in the hot furnace; ran 20 minutes. The flux was fully fluid at the end of the run.

The recovery was as follows: Ingot, 2.1 pounds, or 70 per cent; large buttons, 0.25 pound, or 8 per cent; total yield, 2.35 pounds, or 78 per cent, or 92 per cent on the basis of metal content of charge.

*Run No. 65.*—Three pounds of borings, mixed with 1.5 pounds of flux, was charged in the hot furnace and 1 pound of flux spread on top; ran 20 minutes. The flux was nicely pasty.

The recovery was as follows: Primary yield, 2.2 pounds, or 73.5 per cent; large buttons, 0.05 pound, or 1½ per cent; total yield, 2.25 pounds, 75 per cent, or 88 per cent on the basis of metal content of charge.

*Run No. 66.*—Three pounds of borings were used. No flux was added till the end, then 0.15 pound of sal ammoniac was added and stirred in; the run lasting 17 minutes. The flame used was the same as in previous runs with other fluxes. The melt was only stirred occasionally; very decided glowing or thermit-like action was obtained. The dross was skimmed out on an iron plate and dumped into water at short intervals during skimming.

In this run 1.15 pounds ingot was obtained and many small buttons mixed with adhering dross, which were remelted and yielded 0.1 pound of ingot, 0.05 pound of large buttons; the total yield being 1.3 pounds. The primary yield was 38.5 per cent; the secondary yield, 5 per cent; and the total, 43.5 per cent, or 51 per cent on the basis of metal content of charge.

*Run No. 67.*—Three pounds of borings were used, and no flux was added until the end of the melt, when 0.15 pound of  $\text{ZnCl}_2$  was added and stirred in. The flame was on 12 minutes, the charge being stirred all the time and the borings worked over with a skimmer. The flame was kept as strongly reducing as possible, almost smoky, being much less oxidizing than in previous runs. Not much thermit action was observed until the borings were nearly all melted down, when the whole mass started to heat up almost instantly. The flame was cut off and the mass vigorously stirred for 3 minutes more, then  $\text{ZnCl}_2$  was added. The dross was skimmed off as in run 66.

The recovery was as follows: Primary yield, 1.1 pounds, or 37 per cent; large buttons, 0.35 pound, or 11.5 per cent; total yield 1.45 pounds, or 48½ per cent, or 57 per cent on the basis of metal content of charge.

*Run No. 68.*—The whole charge, consisting of 3 pounds of borings with three-eighth pound  $\text{NaCl-CaF}_2$  flux mixed in, and three-eighth pound spread on top was charged in the hot furnace; ran 16 minutes. The flame was not smoky. The flux and dross were too dry.

The recovery was as follows: Ingot, 1.6 pounds, or 53½ per cent; large buttons, 0.25 pound, or 8½ per cent; total, 1.85 pounds, or 62 per cent, or 73 per cent on the basis of metal content of charge.



When the runs are tabulated in the order of the amount of NaCl-CaF<sub>2</sub> flux used, the results may be summarized as follows:

*Summary of results in tests with reverberatory furnace.*

Run No.	Flux used.	Quantity of flux, based on weight of chips.	Method of adding flux.	Yield of metal—			
				On basis of weight of chips.			Total yield on basis of metal content of charge.
				Primary.	Secondary.	Total.	
		<i>Per cent.</i>		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
66....	NH <sub>4</sub> Cl..	5	At end of melt.....	38.5	5	43.5	51
67....	ZnCl <sub>2</sub> ...	5	...do <sup>a</sup> .....	37	11.5	48.5	57
68....	85 parts NaCl, 15 parts CaF <sub>2</sub> .	25	Half mixed in charge, half placed on top.	53.5	8.5	62	73
58....	...do....	b 25	All mixed in charge c.....	40	31	71	83.5
60....	...do....	33	All placed on top.....	52	6.5	58.5	69
61....	...do....	50	Two-thirds mixed in charge, one-third placed on top.	70	1.5	71.5	84.5
63....	...do....	50	All mixed in charge <sup>d</sup> .....	58	3	61	71.5
59....	...do....	60	One-half mixed in charge, one-half placed on top.	75	1.5	76.5	90
62....	...do....	66	All mixed in charge.....	70	1.5	71.5	84.5
65....	...do....	83	Three-fifths mixed in charge, two-fifths placed on top.	73.5	1.5	75	88
64....	...do....	100	All mixed in charge.....	70	8	78	92

<sup>a</sup> Reducing flame, charge stirred constantly.

<sup>b</sup> 37 per cent including flux used on remelt.

<sup>c</sup> 4 pounds borings, not heated hot enough, 50 per cent of flux used on remelt of secondary yield.

<sup>d</sup> 5 pounds borings, charge too large, causing flame to strike it.

#### REMARKS ON RESULTS OF TESTS.

In run 68 evidently not enough flux was used, as the flux plus dross was dry, and the primary yield was low; that is, there was poor separation.

Run 58 showed even poorer separation, as the temperature was not raised high enough. Increasing the proportion of flux on the remelt made up for this somewhat.

Run 60 also shows a poor primary yield, the flux being all on top and not properly protecting the borings.

In run 63 the charge was too large, so that the flame played directly on much of the surface of the charge.

On account of the greater surface in the reverberatory than in the crucible, it is natural that a greater proportion of flux be required.

Runs 59, 61, 62, 64, and 65, with enough flux, of which a sufficient amount was mixed in, averaged a primary yield of 70 per cent and a total yield of 74.5 per cent, based on the weight of chips, or 87.5 per cent on the basis of metal content, as compared with a primary yield of 40 per cent and a total yield of 50 per cent, or 60 per cent on the metallic basis when NH<sub>4</sub>Cl or ZnCl<sub>2</sub> was used for flux. That is, the tests, though on too small a scale to be conclusive, indicate that with a suitable flux to coat the globules from the melting of the tiny borings, reverberatory melting may be feasible and give as good results

as crucible melting, whereas without such a flux reverberatory melting is undoubtedly not as good as crucible melting.

These tests, like all the previous ones, have shown better experimental results on the same lot of borings when the  $\text{NaCl} - \text{CaF}_2$  flux was used than by any other of the methods advocated.

Robinson <sup>1</sup> has recently obtained a patent on a process of recovering metals in which he melts salt in a reverberatory furnace, then from time to time adds the aluminum-bearing refuse, mixed with 10 per cent of its weight of cryolite, to dissolve aluminum oxide, preferably operating with constant puddling. As this patent appeared after the experiments herein described were completed, no tests were made of this method.

#### ELECTRIC MELTING.

A further test of the  $\text{NaCl} - \text{CaF}_2$  flux was made on a rather larger scale with a crucible electric furnace of the Hoskins type. Runs 17 to 21 had not indicated any appreciable advantage from melting in the reducing atmosphere of this type of furnace, and the furnace was used merely because it was the most convenient one at hand in the laboratory which would take larger charges than the small gas furnace.

#### EXPERIMENTS IN MELTING INGOT.

In three runs on ingot, in which a total quantity of 69 pounds of clean No. 12 ingot was remelted in this furnace without flux, the clean ingot poured weighed 68.5 pounds, also 0.5 pound of skimmings and spillings were obtained, of such metal content that at least 0.3 pound of clean metal would have been obtained on remelting them. That is, the actual metal loss was not over the one-third of 1 per cent given by good melting practice with the iron-pot furnace.

#### EXPERIMENTS IN MELTING BORINGS AND SKIMMINGS.

Three preliminary runs A, B, and C, were made on some odd lots of borings that had accumulated from the previous runs, hereinafter reported, on sizing the borings, and on some skimmings and spillings.

Run A was made on 16.65 pounds of miscellaneous borings, different from but approximately of the same quality as those used in run B, with which 20 per cent by weight of the flux consisting of 85 per cent  $\text{NaCl}$  and 15 per cent  $\text{CaF}_2$  was mixed. The recovery, based on total weight of chips taken, was 11.8 pounds ingot, 0.6 pound buttons, or a primary recovery of 71 per cent and a total recovery of 74.5 per cent.

In run B, on 15 pounds of miscellaneous borings plus 20 per cent of the flux, the primary yield was 11 pounds ingot, or 73.5 per cent, and the total yield 12.2 pounds, or 81.5 per cent, on the basis of total weight of chips.

---

<sup>1</sup> Robinson, C. S., U. S. Patent 1180435, April 25, 1916.

In run C, on 13.5 pounds skimmings, plus 20 per cent flux, recovery was 11 pounds ingot, or 11.25 pounds in all, or a primary yield of 82 per cent and a total yield of 83.5 per cent. Two runs were then made on chips from lot B with the following results:

*Results of melting miscellaneous borings with  $\text{NH}_4\text{Cl}$  and  $\text{NaCl-CaF}_2$  flux.*

	Run No. 69.	Run No. 70.
Observers.....	G, L	G, L
Weight of chips taken, pounds.....	15	15
Flux used.....	$\text{NH}_4\text{Cl}$	85 parts $\text{NaCl}$ . 15 parts $\text{CaF}_2$ .
Percentage of flux used, on basis of weight of chips	2	20
Yield of metal:		
Primary yield, percentage of weight of chips.	53	68.5
Total yield, percentage of weight of chips...	58	71.5
Total yield, percentage of metal content of charge.....	68.5	84

As the crucible was loose in the furnace, not being supported at the sides, puddling was not feasible, so in run 69 the borings were poked and stirred quite constantly as long as oil was smoking off, but the metal was allowed to become more fluid between chargings than was the case in the puddling experiments.

In run 70 no stirring was done till the end, when the temperature was raised well above  $900^\circ \text{C}$ ., so that the flux was really fluid.

#### COMMERCIAL TEST OF SALT AND FLUORSPAR FLUX.

As the iron pot, puddling process and the crucible, salt and fluorspar process appeared to be, from the experimental results and from the reports of the users of the different processes, the two most satisfactory in commercial use, it was suggested to the firm using the puddling process that to test that process against the salt and fluorspar process on the same lot of borings would be of interest. It was pointed out that the higher temperature required in the latter process to make the flux fluid enough to allow a good separation might be too high to permit practical life of the iron pot.

This firm made such a test, using an iron-pot furnace for both methods, and reported as follows:

A lot of borings that had been passed through a magnetic separator was carefully weighed, mixed, and divided into two parts. One part was melted down by puddling, about 2 per cent of zinc chloride being used as final flux; the other part was mixed with 25 per cent of a flux consisting of 85 per cent salt and 15 per cent fluorspar. The cost of flux per 100 pounds of chips was 10.4 cents for zinc chloride and 10.8 cents for the salt and fluorspar flux. The chips contained 13.3 per cent oil and 0.6 per cent free iron left after commercial magnetic separation. The dross from each melt was put in water, concentrated in a tumbling barrel, dried, and assayed, by running down in a small crucible, for its metal content.

*Test 1.*—1,049 pounds of chips plus 20 pounds of zinc chloride yielded 778 pounds of ingot and 27 pounds of concentrated skimmings containing 74 per cent metal, or 20 pounds, a total of 798 pounds, giving a primary recovery of 74.16 per cent and a total recovery of 76.06 per cent.



*Test 2.*—508 pounds of chips mixed with 126 pounds of the flux consisting of 85 per cent salt and 15 per cent fluorspar yielded 376 pounds ingot and 22 pounds concentrated skimmings of 62.5 per cent metal content, or 14 pounds, a total of 390 pounds, giving a primary recovery of 74.01 per cent and a total recovery of 76.71 per cent.

Test 1 took 6½ hours and test 2 took 8 hours, or the ingot produced per hour was 119 pounds with the puddling method as compared with 47 pounds by the salt and fluorspar method; that is, the production was cut down 60 per cent. The iron pot used in test 2 was practically ruined at the end of the test, owing to the high temperature necessary to melt the flux. The metal is heated too hot during the melting, thus injuring the quality, in the salt and fluorspar method. The results show that the salt and fluorspar flux can not be used with an iron pot, for the flux has little effect if not pasty or molten, and the temperature necessary for this result is too high for the iron pot. The best place to use such a flux is in melting grindings or very dirty borings. When work slackens somewhat we are going to make a test with a crucible, and will inform you of the results.

No report has yet (June, 1916) been received on the crucible test. From some other comments it seemed possible that this firm had used continuous puddling on test 2 as well as on test 1, although in suggesting the test it was pointed out that continuous stirring was apparently not desirable with the salt and fluorspar flux. An inquiry on this point, and as to whether the borings contained dirt as well as oil was unanswered. From the comment on the apparent applicability of the salt and fluorspar flux to dirty borings, it would seem probable that the borings tested were not dirty.

If the true metal content be taken as the oil-free content, the recovery of 76 to 77 per cent obtained by both methods is equivalent to 88 to 89 per cent on the basis of true metal content, or well above the 68 to 70 per cent recovery, on the basis of original weight of chips, reported by this firm as its average yearly recovery on all borings melted by the puddling process, which would indicate that the borings used were of better quality than the average.

Although the results of this test are unfavorable to the use of salt and fluorspar flux for iron-pot melting, they do not show how puddling in an iron pot and using the salt and fluorspar flux in coke or oil-fired crucible furnaces would compare. The output per furnace was less with the salt and fluorspar flux, but it should be remembered that when salt and fluorspar flux is used the charge does not require continuous stirring, as in the puddling process, and that the output per furnace tender should therefore not be reduced. Of course, the melting of 25 per cent of flux will require more fuel than in a process which does not use a large proportion of flux. Moreover, if the results of the small-scale tests on reverberatory melting would hold on a commercial scale, it would appear that the fuel and labor economies possible from melting in a reverberatory on a larger scale than with crucible or pot furnaces, might perhaps be better gained by using the salt and fluorspar flux, as puddling in a reverberatory would be very difficult.



## EFFECT OF SIZING THE BORINGS.

In order to determine whether sifting the borings and melting the larger and the smaller sizes separately would be of advantage, some borings from lot A were sifted on a 20-mesh riddle, 80 per cent remaining on it and 20 per cent passing through it. The results of runs made on these borings were as follows:

*Results of melting sized borings from lot A with  $\text{NH}_4\text{Cl}$  flux.*

Size of borings.....	Run No. 71.	Run No. 72.
	Over 20-mesh.	Under 20-mesh.
Observer.....	G	G
Weight of chips used, kilograms.....	2	2
Percentage of flux, on basis of weight of borings...	2	2
Yield of metal, on basis of weight of chips:		
Primary yield, per cent.....	50.5	8.4
Total yield, per cent.....	67.5	18.3

As the total yield on the larger borings, which make up 80 per cent by weight of the unsifted borings, was 67.5 per cent and that on the smaller borings, which make up 20 per cent of that weight, was 18.3 per cent, the total yield on the basis of weight of unsifted chips was  $\frac{(67.5 \times 80) + (18.3 \times 20)}{100} = 58$  per cent, which is equivalent to 64.5 per cent of the actual metal content.

A 14-mesh riddle was used in order to get a better separation of larger borings from the smaller ones, and the material passing 14-mesh was sifted on a 40-mesh screen to free it from dirt, the material under 40-mesh being discarded. The two sizes were each melted with the  $\text{NaCl-CaF}_2$  flux by heating to  $900^\circ \text{C}$ . The following results were obtained:

*Results of melting sized borings from lot A with  $\text{NaCl-CaF}_2$  and  $\text{ZnCl}_2$ .*

Run No.	Lot.	Weight of charge.	Size of borings.	Observer.	Flux used.	Quantity of flux used, on basis of weight of chips.	Yield of metal, on basis of weight of chips.	
							Primary.	Total.
73....	A	<i>Kg.</i> 1.25	Over 14-mesh.....	J	85 parts $\text{NaCl}$ , 15 parts $\text{CaF}_2$ .	<i>Per cent.</i> 15	<i>Per cent.</i> 84	<i>Per cent.</i> 84.5
74....	A	1.25	....do.....	J	....do.....	15	87	88
75....	A	1.25	....do.....	J	....do.....	25	87	88
76....	A	1.25	....do.....	J	....do.....	25	86	86.5
77....	A	1	Between 14 and 40 mesh.	J	....do.....	25	71	72
78....	A	1	....do.....	J	....do.....	25	73.5	74
80....	A	1	....do.....	J	....do.....	50	68	68.5
81....	A	1	....do.....	J	....do.....	50	70.5	71
82 <sup>a</sup> ...	A	1.5	....do.....	J	$\text{ZnCl}_2$ .....	2	56	61.5

<sup>a</sup> Run 82 was made by the puddling process.

In the runs in which the  $\text{NaCl-CaF}_2$  flux was used the average total recovery on the material over 14-mesh was 87 per cent and on that between 14 and 40 mesh it was 71.5 per cent. As 54 per cent of the unsifted borings passed the 14-mesh riddle, 42 per cent was between 14 and 40 mesh, and 4 per cent was under 40-mesh and discarded, being practically all dirt. The results may be figured as follows:

$$\frac{(54 \times 87) + (42 \times 71.5)}{100} = 77 \text{ per cent, which is the total recovery on the}$$

basis of the original weight of dirty borings, or 85.5 per cent on the basis of metal content, a recovery that is only slightly better than the average of results on unsifted material and is the same as the maximum. On the very fine material the use of salt and fluorspar flux gave markedly better results than puddling, which suggests that the borings might be sifted and the puddling process used on the cleaner and larger fraction, thus allowing the use of the iron pot for most of the chips, and the smaller and dirtier part melted in crucibles in coke or oil fires with  $\text{NaCl-CaF}_2$  flux. This method would reduce the fuel consumption, as more would be required to melt the large amount of  $\text{NaCl-CaF}_2$  flux needed were it to be used on all of the borings without sifting them.

Some chips from lot B were also sifted on 14 and 40 mesh riddles, giving 57.5 per cent over 14-mesh, 30.3 per cent between 14 and 40 mesh, and 12.2 per cent under 40-mesh, which was practically all dirt, and discarded. A run (No. 83) was made on the material that was over 14-mesh, in the electric furnace used for runs 69 and 70. The results were as follows:

*Result of melting chips from barrel B, over 14-mesh in size, in an electric furnace (run No. 83.)*

Observers, G and L.

Weight of chips used, 15 pounds.

Flux used, 85 per cent  $\text{NaCl}$  and 15 per cent  $\text{CaF}_2$ .

Quantity of flux used, based on weight of chips, 20 per cent.

Yield of metal, on basis of weight of chips:

Primary yield, 86.5 per cent.

Total yield, 89 per cent.

This larger scale test on the material over 14-mesh from lot B agreed well with the tests in the small gas furnace (runs 73 to 76) of the material over 14-mesh from lot A. The material from 14 to 40 mesh in size was used in washing tests.

As the large chips are much easier to handle so as to get a high recovery than those under 20-mesh, it would pay to separate them for remelting if some use could be found for the fine material. Where, as in the case cited on page 10, there is a demand for chips for chemical use, this could be done. Possibly the fine material might be used in calorizing, after further comminution, and would offer a cheaper

source of aluminum for that purpose than the aluminum powder now used, provided the copper or zinc in the alloy chips was not detrimental, and provided also that the dirt in the chips would take the place of the aluminum oxide now added to prevent coalescence.

#### EXPERIMENTS IN WASHING THE CHIPS.

The results in all of the experiments described go to show that clean borings may be melted with comparatively little loss, and that a small content of fine dirt greatly increases the difficulty of making the tiny globules coalesce. When the borings are oily, the dirt sticks tightly to them, so that simply sifting them will not remove the dirt.

Obviously the next experiment to try, then, was to wash the borings so as to free them from oil and then sift out the dirt.

#### WASHING WITH HOT WATER.

An attempt was made to wash chips from lot A with hot water, in the hope that the oil would be freed from the chips and float to the top. But although the chips in lot A contained about 3 per cent oil, as was determined by ether extraction, not a trace of oil was floated off by the water. Bancroft<sup>a</sup> has explained this retention of oil by the fact that aluminum adsorbs oil more readily than it does water. He has found in flotation experiments that if aluminum powder be shaken with kerosene and water, the aluminum goes entirely into the kerosene layer, being floated up by it. In the experiment by the writer the chips, being too heavy to be floated by the small amount of adsorbed oil, held the oil down with them instead of rising with it as in the case of the powder in the presence of much oil.

#### WASHING WITH A WATER-SOLUBLE CUTTING COMPOUND.

Inasmuch as a high recovery was reported on chips cut with a water-soluble cutting compound and then centrifuged, it was thought possible that oily borings washed with this compound might be cleaned and freed from dirt, so some of the compound was obtained through the courtesy of the makers.

This compound was a stiff white emulsion of oil, soda soap, and water, the water forming over half of the material. On ignition it formed 2 to 2.5 per cent ash, mainly  $\text{Na}_2\text{CO}_3$ . It was practically neutral in reaction, showing a very faint alkaline reaction with phenolphthalein, due to hydrolysis of the soap. When the compound was stirred into water a very stable milk-white emulsion was formed.

---

<sup>a</sup> Bancroft, W. D., In discussion. Trans. Am. Electrochem. Soc., vol. 23, 1913, p. 294.

Washing chips from lot A with this emulsion did not remove appreciable amounts of oil or dirt, so that the high recovery reported seems due to cleanliness rather than to any specific action of the cutting compound.

#### WASHING WITH GASOLINE.

Washing with gasoline was then attempted. The borings were washed on a 20-mesh riddle and the fraction over 20-mesh and that under 20-mesh were dried separately. The dry material under 20-mesh was then sifted on an 80-mesh riddle. Only one change of gasoline was used, but it took nearly 5 gallons of gasoline to wash about 60 pounds of borings. The results of washing were as follows: Oil eliminated, 3 per cent; dirt eliminated, 7 per cent; clean borings over 20-mesh, 72 per cent; clean borings under 20-mesh, 18 per cent.

The borings over 20-mesh were quite bright, but there was still some oil not yet washed out and some dirt adhering to them, and on melting down these borings oil still smoked off, though the smoking stopped much sooner than with the original unwashed material. The washed borings under 20-mesh contained a little more oil than those over 20-mesh, and considerably more dirt. As no accurate method of analysis to determine the amount of dirt still retained could be devised, the assumption was made for purposes of computation that the borings after washing were 100 per cent metal.

A heat of washed borings was then run with a mixture of 72 parts of borings over 20-mesh and 18 parts of borings under 20-mesh—that is, a mixture that could be considered as washed but not sifted. The results of this run were as follows:

#### *Results of melting borings washed in gasoline (run No. 83).*

Chips from lot A used in test.

Observer, G.

Weight of chips used, 2 kilograms.

Flux used,  $\text{NH}_4\text{Cl}$ .

Quantity of flux used, based on weight of chips, 2 per cent.

Yield of metal, on basis of weight of chips:

Primary yield, 60 per cent.

Total yield, 67 per cent.

As this charge weighed but 90 per cent of what the unwashed borings would weigh, and the washed borings are assumed to be all metal, the total yield given, 67 per cent, is on the basis of metal content, and the total yield on the basis of original weight of chips is 60 per cent.



Samples of the two fractions were then run separately, with results as follows:

*Results of melting the fractions separately.*

	Run No. 84.	Run No. 85.
Weight of chips used, kilograms.....	.....	.....
Observer.....	G.	G.
Size of borings.....	Over 20-mesh.	Under 20-mesh.
Flux used.....	NH <sub>4</sub> Cl	NH <sub>4</sub> Cl
Percentage of flux used on basis of weight of chips.....	2	2
Yield of metal, on basis of weight of chips:		
Primary yield, per cent.....	73.5	26.8
Total yield, per cent.....	81.5	37.3

The average total yield, on the basis of the original weight of unsifted borings, is  $\frac{(72 \times 81.5) + (18 \times 37.5)}{100} = 65.5$  per cent, or on the basis of true metal content is 73 per cent. Samples of the two fractions were also run separately, poking each into a heel of 1 kilogram of molten metal, with the following results:

*Results of melting borings with heel of molten metal.*

Run No.	Chips used.	Charge used.	Size.	Observer.	Flux used.	Quantity of flux used on basis of weight of charge.	Yield of metal on basis of weight of charge.		
							Primary.	Total.	Average.
86....	Chips from lot A, washed with gasoline.	Kg. 1	Over 20-mesh.	G.....	NH <sub>4</sub> Cl..	Per cent. 2	Per cent. 70	Per ct. 80.5	.....
87....	.....do.....	1	Under 20-mesh.	G.....	.....do.....	2	29.5	41.5	} 40.5
88....	.....do.....	1	.....do.....	G.....	.....do.....	2	23.5	39.5	

By this method the total yield, on the basis of weight of unsifted borings, is  $\frac{(72 \times 80.5) + (18 \times 40.5)}{100} = 65$  per cent, or on the basis of true metal content is 73 per cent.

When runs 84 to 88 are compared with runs 2 to 7 and 24, 25, in which similar melting methods were used on unwashed, unsifted material it appears that there is probably a slight advantage, as regards recovery, in washing and sifting.

Some of the material over 20-mesh washed with gasoline was melted with the NaCl-CaF<sub>2</sub> flux, with the following results.

*Results of melting washed borings over 20 mesh in size with NaCl-CaF<sub>2</sub> flux.*

	Run No. 89.	Run No. 90.
Observer.....	J.	J.
Size of borings.....	Over 20- mesh.	Under 20- mesh.
Weight of charge used, kilograms.....	1. 25	1. 25
Percentage of flux used, based on weight of borings.....	25	25
Yield of metal, on basis of weight of borings used:		
Primary yield, per cent.....	86. 5	87
Total yield, per cent.....	87. 5	88

The results check with those obtained on melting unwashed borings over 14-mesh with this flux in runs 73 to 76 on chips from lot A and in run 83 on chips from lot B, but are not superior, so that if washing is of advantage its advantage is mainly on the finer borings.

#### WASHING WITH CAUSTIC-SODA SOLUTION.

No runs were made with this flux on the fraction under 20-mesh, washed in gasoline, as the supply of this fine material was exhausted and gasoline washing seemed to be too expensive a process for commercial use. However, some of the material between 14 and 40 mesh left from the sizing of part of lot B for run 83 was washed in a dilute solution of caustic soda. Caustic soda or caustic potash in dilute solution, is used as a cleaner before polishing aluminum articles. If saponifiable oil or grease is present, it will form a soap that will tend to emulsify paraffin oil. If there is no saponifiable matter in the cutting oil, soap could be added to the NaOH or KOH solution.

Preliminary tests showed that the oil and dirt in chips of lot B were so loosened by the action of a 0.5 per cent solution of NaOH that if the chips were placed in the solution, allowed to remain with constant stirring till slight chemical attack of the NaOH on the aluminum had begun, as shown by a fairly rapid evolution of gas bubbles, a great deal of the dirt was brought to the top by the gas and could readily be floated off with water, which also serves to wash out the caustic solution.

After the chips had been washed, they were freed from water by centrifuging and then drying at 100° C.

The method of washing was rather crude, 1,500 grams of the chips between 14 and 40 mesh being placed in a pan, 5 liters of 0.5 per cent NaOH solution added, and the whole stirred till there was a fairly copious evolution of gas, which required about 10 minutes. Water was then run into the bottom of the pan through a tube and allowed to overflow with constant stirring of the chips until the water ran clear, which took about 10 minutes more.

The chips were then drained and centrifuged. The only centrifuge available was of very small capacity and there was a slight

mechanical loss in transferring the chips. From 1,500 grams of the 14 to 40 mesh chips, 1,412 grams of slightly damp chips were obtained after centrifuging, which, after being dried at 100° C. and 15 grams of dirt sifted out on an 80-mesh screen, weighed 1,362 grams—that is, 9 per cent of the 1,500 grams of the 14 to 40 mesh fraction, or 3 per cent of the unsifted dirty borings, was removed from this fraction by washing. The 1,362 grams were then run down with 20 per cent of the NaCl-CaF<sub>2</sub> flux.

The results of the test were as follows:

*Results of melting chips 14 to 40 mesh in size, washed in NaOH solution (run No. 89).*

Chips used, from lot A, 14 to 40 mesh, washed in NaOH solution.

Observer, J.

Weight of chips used, 1.33 kilograms.

Flux used, 85 per cent NaCl, 15 per cent CaF<sub>2</sub>.

Quantity of flux used, based on weight of charge, 20 per cent.

Yield of metal, on basis of weight of chips:

Primary yield, 72 per cent.

Total yield, 75.5 per cent.

On the basis of the 1,500 grams originally taken, the total recovery was 68.5 per cent, which is slightly less than the yield of 71.5 per cent obtained in runs 77 to 81, which were made on the unwashed 14 to 40 mesh fraction from lot A, but it must be remembered that the chips in lot A were not so dirty as those in lot B.

As the recovery on the unwashed fraction over 14 mesh in size used in run 83 was 89 per cent, the total recovery, calculated on the basis of original weight of unsifted material, is

$$\frac{(57.5 \times 89) + (30.3 \times 68.5)}{100} = 72 \text{ per cent,}$$

or, on the basis of true metal content, 85 per cent.

Two 1,500-gram samples of the unsifted borings from lot B were then washed with NaOH solution, centrifuged, and dried, one yielding 1,326 grams and the other 1,316 grams. These samples were run down with 20 per cent of NaCl-CaF<sub>2</sub> flux, with results as follows:

*Results of melting unsifted borings from lot B, washed in NaOH solution.*

	Run No. 90.	Run No. 91.
Observer.....	J	J
Weight of borings used, kilograms.....	1.5	1.5
Flux used.....	80 parts NaCl 15 parts CaF <sub>2</sub>	80 parts NaCl 15 parts CaF <sub>2</sub>
Quantity of flux, on basis of weight of chips, per cent.....	20	20
Yield of metal:		
Primary yield, percentage of weight of chips.....	71	70
Total yield, percentage of weight of chips.....	74.5	75
Total yield, percentage of metal content of chips.....	87.5	88.5

These results show enough improvement over those on unwashed chips from lot B to make it seem possible that washing dirty borings with dilute NaOH solution might be worth while. One method that suggests itself for trial on dirty borings is to use a concentrating table of the Willfley type, fed not with water but with dilute NaOH solution, or with a solution of both NaOH and soap, the solution to be run into a settler and pumped back into the system, followed by washing with water and centrifuging.

From the comparative readiness with which dirt was freed from the chips in the experiments it would seem that such treatment would be worth a commercial test. As most refining plants and many nonferrous foundries are now equipped with concentrating tables for use on brass-foundry wastes, the only unusual piece of apparatus required would be the centrifuge.

Borings thus cleaned and dried would probably be more readily separated from free iron on being passed over the magnetic separator than oily borings. Were a centrifuge installed, very oily borings, such as contain 10 per cent or more of oil, might be run through the centrifuge before washing, which would leave less work for the cleaning solution to do, and the cost of the operation would probably be compensated for by the value of the oil recovered.

#### MISCELLANEOUS EXPERIMENTS IN ELECTRIC FURNACES.

After the experiments above cited had been completed, another indirect arc furnace was constructed for other purposes, and the question arose whether or not the furnace would be applicable to melting aluminum chips.

Another lot of No. 12 alloy chips, designated as lot D, was obtained from a large refining company, which stated that it had made a 77 per cent recovery, on the basis of the weight of the chips, from these chips by the puddling process.

On chips that were sized without extracting the oil, the recoveries were as follows: Chips over 14 mesh in size, 62 per cent; between 14 and 40 mesh, 30 per cent; and under 40 mesh, 8 per cent. By extraction with ether, 5 per cent of oil was obtained; and by sifting the oil-free borings on a 40-mesh sieve, and carefully floating off the dirt from the small amount of aluminum that passed the 40-mesh sieve, 7 per cent of dirt was obtained. Hence the chips contained at least 12 per cent of nonmetallic material, and the metal content was not over 88 per cent. This means that in commercial practice the puddling process gave a recovery of 87.5 per cent of the metallic content.



## EXPERIMENTS WITH INDIRECT ARC FURNACE.

The indirect arc furnace resembled in form a horizontal cylinder. The melting chamber was 15 inches in diameter and about 15 inches long. Graphite electrodes 2 inches in diameter were passed horizontally into the center of the chamber, and the arc was formed between them. The furnace lining forming the melting chamber was of fire brick, coated with asbestos cement, and this layer was coated with alundum magnesite mixture.

## MELTING WITH ALUNDUM-MAGNESITE LINING.

*Run No. 92.*—Twenty-five pounds of chips, mixed with 10 pounds of flux, consisting of 85 parts NaCl and 15 parts  $\text{CaF}_2$  was charged into the cold furnace. The charge came within about 2 inches of the arc. After 42 minutes heating in which 20 kilowatt-hours were used, a pool of metal had collected in the bottom, and the temperature was  $900^\circ\text{C}$ . The flux was not fluid and still held many buttons of metal, and a few unmelted chips remained in the bottom of the furnace. The yield of ingot was 6.25 pounds, or a primary yield of 25 per cent on the basis of original weight of chips charged. The flux was scraped out and on being crushed and washed yielded 8.25 pounds of buttons, or 35 per cent, which, with the 25 per cent previously obtained, made a total recovery of 60 per cent, on the basis of original weight of chips.

It was evident that the charge as a whole was not hot enough, and that it would have been better not to charge so many chips at once. The magnesite-alundum lining was badly attacked and began to crumble off, and as this was believed to be due to the effect of the fluorspar, it was decided to try common salt in the next heat.

*Run No. 93.*—Ten pounds of chips mixed with 5 pounds NaCl was charged into the cold furnace and the current turned on, 17 kilowatt-hours being used in 29 minutes, then 10 pounds of chips mixed with 5 pounds of NaCl was added. Heating was continued 25 minutes more, using 13 kilowatt-hours, or the total power input was 30 kilowatt-hours. The temperature was over  $1,000^\circ\text{C}$ , but the mass of NaCl and dross was only pasty. From the 20 pounds of chips charged, 7.5 pounds ingot and 5 pounds of buttons were obtained, or a primary yield of 37.5 per cent and a total yield of 62.5 per cent, on the basis of original weight of chips. The lining continued to crumble.

*Run No. 94.*—The furnace was preheated by the use of 15 kilowatt-hours and then 15 pounds of chips, plus 15 pounds of flux, consisting of 85 parts NaCl and 15 parts of  $\text{CaF}_2$ , was charged. Ten pounds of the flux was mixed with the chips, and 5 pounds placed on top of the charge. After 27 minutes, in which 15 kilowatt-hours more

was used, the charge was at a temperature of  $900^{\circ}\text{C}$ . The bulk of the flux was fluid, but it contained some matted masses of flux and dross that were only pasty. Seven and one-half pounds of ingot and some flux were poured. The flux that poured out with the metal was replaced in the furnace and heated 20 minutes, in which 10 kilowatt-hours were used. Two and nine-tenths pounds ingot was obtained on the second pour. The flux was all fluid at this time. Besides the total of 10.4 pounds of ingot, 0.9 pound of buttons was recovered from the slag, giving a primary or ingot yield of 69.5 per cent and a total yield of 75 per cent on the basis of the weight of chips charged. The lining was still crumbling badly, and after the furnace had cooled all loose pieces were taken out. By this time most of the alundum magnesite layer of the lining had come off. It appeared that with a lining not attacked by the flux and with sufficient flux fair recoveries might be made.

*Run No. 95.*—In order to determine whether less flux than was used in the previous run would serve, run 95 was made, after the furnace had been preheated for 15 kilowatt-hours, by charging 18 pounds of borings mixed with 8 pounds of the flux (85 parts NaCl, 15 parts  $\text{CaF}_2$ ) and placing 3 pounds of flux on top of the charge. After 44 minutes, in which 22.5 kilowatt-hours was used, there was a pool of metal in the furnace, and 10.2 pounds of ingot was poured, but the flux was not fluid. The heat was continued for 18 minutes, 7.5 kilowatt-hours being used, and 1.4 pounds more ingot was poured. The flux was still only pasty. Five-tenths pound of buttons was recovered from the slag, giving a combined primary or ingot yield of 64.5 per cent and a total yield of 67.5 per cent on the basis of chips charged.

The furnace was then cleaned, and a run made on a charge of 37 pounds of ingot. Starting with a cold furnace, it took 52 minutes' time and a power input of 30 kilowatt-hours to heat the charge to  $735^{\circ}\text{C}$ . Twenty-five pounds more of ingot was then added, and 10 more kilowatt-hours used in 13 minutes. The charge was then at  $870^{\circ}\text{C}$ ., or hotter than is needed for pouring aluminum castings. Of course, after the furnace is once hot, the power needed to melt a given charge will be less than when the furnace must be heated also. Moreover, a furnace of commercial size will be much more efficient than a small laboratory furnace. Nevertheless, the power required for the electric melting of aluminum, even under the most favorable conditions, will be high. In this heat, which was started with a cold furnace, a total power input of 40 kilowatt-hours was required to heat 62 pounds of aluminum to  $870^{\circ}\text{C}$ ., and 10 kilowatt-hours to melt 25 pounds and raise it to  $870^{\circ}\text{C}$ ., and also to raise the first 37 pounds from  $735^{\circ}$  to  $870^{\circ}\text{C}$ . This furnace, started cold, has melted 125

pounds of red brass ingot and heated it to 1,150° C. with 40 kilowatt-hours, and after the furnace was hot 19 kilowatt-hours has melted and heated to 1,200° C. 130 pounds of red brass ingot.

That is, it took about twice as much power per pound to heat the aluminum to its pouring temperature as it did to heat red brass to its pouring temperature. This is in accord with theory. Richards <sup>a</sup> has calculated that theoretically about 278 heat units are required to heat aluminum to about 710° C., whereas only about 130 heat units are required to heat brass or bronze to about 1,050° C. The difference is due to the extremely high heat of fusion of aluminum, the relative heats of fusion being as follows: Aluminum, 100; copper, 43; zinc, 23; tin, 14; and lead, 4. This high heat of fusion overbalances the higher temperatures needed for brass and bronze.

Only in exceptional circumstances can electric furnaces compete with fuel-fired furnaces in melting brass, unless the alloy is one on which the use of an electric furnace can reduce the zinc loss, because of the high cost of electric power compared to fuel. Therefore it is evident that there is little hope of the electric furnace being able to compete with the oil-fired, iron-pot furnace in melting ingot aluminum, as ingot can be melted in pot furnaces with a loss of only one-third of 1 per cent, and there is no such margin for metal saving as in the case of brasses high in zinc.

When a large proportion of flux is to be melted, as in the fluxing method for aluminum chips, still more power will be required than in the melting of aluminum ingot. In the experimental furnace, and in most tilting electric furnaces, it is inconvenient or impossible to press the charge of chips and flux down into a compact mass, as is done in crucible melting. It is well known <sup>b</sup> that at temperatures below a red heat, a porous mass with inclosed air spaces is a good heat insulator and a poor heat conductor, so that notwithstanding the high heat conductivity of aluminum a mass of chips does not conduct heat very rapidly. Thus, if the layer of loose chips on the furnace hearth is thick, the top part of the layer, being nearest the arc, may be seriously overheated before the bottom part is hot enough to melt. From the salt fumes coming from the arc furnace when pouring, it appears probable that much salt is volatilized from the top of the layer, and that the chips on the top of the bath fail to coalesce and are so hot that they oxidize as soon as the furnace is opened to charge or to pour, and air enters. Hence much more flux is necessary than where the borings can be jammed down into a crucible, although in crucible melting with flux the constant stirring necessary in the puddling process is not required.

<sup>a</sup> Richards, J. W., Electric power required to melt metals: Trans. Am. Brass Founders' Assn., vol. 4, 1910, p. 93.

<sup>b</sup> Ray, W. T., and Kreisinger, H., The flow of heat through furnace walls: Bull. 8, Bureau of Mines, 1911, 32 pp.



In an arc furnace the main source of heat is the arc itself, although some heat is of course reflected from the roof. As the arc is much hotter and much smaller than a flame, there is more danger of local overheating at the surface of the charge in an arc furnace than in a reverberatory furnace, where a sheet of flame covers the whole charge. An uncompacted mass of chips and flux, even at  $900^{\circ}\text{C}$ ., is so viscous that rocking the arc furnace back and forth fails to stir the mass enough to mix it and prevent local overheating.

Hence, although there is a possibility that in a suitably lined electric furnace, where the lining is not attacked by the flux (linings of magnesite brick or mixtures of graphite and fire clay might perhaps serve) aluminum borings could be melted on a larger scale than in iron pots or crucibles, the reduction in labor cost would probably be more than offset by the higher cost for power as compared with the cost for fuel in ordinary types of furnaces.

#### EXPERIMENT WITH GRAPHITE CRUCIBLE LINING.

Another run was made to determine whether the indirect arc furnace would give satisfactory results with a better lining, as regards losses, without using so much flux as in run 94. It was thought that the lining that had crumbled off had acted like so much dirt in the chips and required excessive amounts of flux.

As neither magnesite brick nor graphite and fire clay mixture was at hand, a No. 60 crucible was used as a lining for the furnace. A hole for the passage of the electrode was cut in the bottom of the crucible, and a hole for the charging door and pouring spout was cut in the side. The crucible was placed horizontally inside the furnace and fire clay was packed between the crucible and the former furnace lining and the hearth at the mouth end of the crucible, the packing being completed with a mixture of ground crucibles and enough fire clay for a bond. This reduced the capacity of the furnace very greatly.

*Run No. 96.*—After the furnace had been preheated with a power consumption of 12 kilowatt-hours, about two-thirds of a charge of 8 pounds of chips mixed with 2.4 pounds of flux consisting of 85 parts NaCl and 15 parts  $\text{CaF}_2$  was added. After 15 minutes, when 8 more kilowatt-hours had been used, the other third was charged, and after heating 5 minutes longer, 3 kilowatt-hours being used, the charge was at  $975^{\circ}\text{C}$ . The flux was only pasty and some thermit-like action was noted on top of the charge. Three and seventy-five hundredths pounds ingot and 1.25 pounds buttons were obtained, or a primary yield of 47 per cent, and a total yield of 62.5 per cent, on the basis of chips charged. As the crucible used as lining was not attacked and the result of this run was poor, the poor results of previous runs were evidently not mainly due to the crumbling of the lining.



Although run 94 indicated that with a sufficient amount of flux, electric melting in an indirect arc furnace might be possible, it seemed very unlikely that it would be commercially profitable and experiments with the indirect arc furnace were discontinued.

#### EXPERIMENTS WITH CRUCIBLE, RESISTOR FURNACE.

As lot D was from chips on which the commercial recovery was known to be 77 per cent on the weight of the chips, it was decided to make a few crucible melts to compare the experimental results with the commercial ones.

For these melts a No. 10 crucible was embedded in the granular graphite resistor of an electric resistor furnace, the crucible being taken out for pouring.

*Run No. 97.*—The crucible was first heated to dull redness, then small amounts of borings were added from time to time and puddled, care being taken not to let the temperature rise appreciably above the melting point. It took 10 minutes thus to work in 3 pounds of borings. Then the crucible was covered and heated 3 minutes more, when 2 per cent of fused  $\text{ZnCl}_2$  was stirred in, the dross skimmed into water, and the metal poured. The dross showed a little thermit-like action. Two and twentth-five hundredths pounds ingot and 0.5 pound buttons was obtained, or a primary yield of 75 per cent and a total yield of 77 per cent, on the basis of the weight of the chips—that is, the same recovery was made as was found commercially.

*Run No. 98.*—Borax, which has been suggested as a flux for aluminum chips, not having been previously tried, 3 pounds of chips and 0.75 pound of pulverized crystalline borax, equivalent to about 0.4 pound borax glass, were mixed together and most of the mixture was charged into the cold crucible. The furnace was then heated and after 16 minutes the rest of the charge was added and the whole poked continually as it melted down, to approximate conditions in the puddling process. After 7 minutes more the charge was heated to more than  $1,000^\circ \text{C}$ . but the borax flux was only pasty, and mostly at the bottom of the crucible. The ingot recovery was 2.1 pounds and there was 0.20 pound of buttons, giving a primary yield of 70 per cent and a total yield of 76.5 per cent on the basis of chips charged. There was no thermit-like action.

*Run No. 99.*—As it seemed possible that the use of a larger amount of borax, with the addition of some fluorspar, might give a more fluid flux, the next run was made as follows: Three pounds of chips mixed with 1.2 pounds of pulverized crystalline borax and 0.1 pound fluorspar were charged into the hot crucible. In 16 minutes the temperature was over  $1,000^\circ \text{C}$ ., but the aluminum was all on top and the flux all on the bottom. Only 1.3 pounds ingot, or 43 per cent primary recovery, was obtained. The mass of flux in the bottom of the crucible was full of exceedingly minute globules of

metal, which were not recovered. Borax is evidently too heavy to serve as a good flux.

*Run No. 100.*—Five pounds of chips was mixed with 1.5 pounds of the flux, consisting of 85 parts NaCl and 15 parts  $\text{CaF}_2$ , and as much as possible of the mixture was packed into the cold crucible. After 20 minutes heating the rest was added, and after a total of 49 minutes the charge was at  $950^\circ \text{C}$ . The flux was quite fluid. Three and seventy-five one-hundredths pounds ingot and 0.10 pound of buttons were obtained, or a primary yield of 75 per cent and a total yield of 77 per cent on the basis of weight of chips taken.

*Run No. 101.*—Five pounds of chips was washed 10 minutes with 5 liters of 0.5 per cent NaOH solution, which removed much dirt, the NaOH was washed out, and the chips were stored under water 4 hours. They were then drained of water, mixed with 1.5 pounds of the flux, consisting of 85 parts NaCl and 15 parts  $\text{CaF}_2$ , and most of the damp mixture charged into the warm crucible. No oil smoked off, showing that it had been removed by the washing. After 15 minutes the rest of the charge was added, and after 15 minutes more the temperature was  $975^\circ \text{C}$ . The slag was fluid. Three and sixty-five one-hundredths pounds of ingot and 0.15 pound of buttons were obtained, or a primary yield of 73 per cent and a total yield of 76 per cent on the basis of weight of chips. There was a slight mechanical loss in washing and transferring the moist chips.

#### CONCLUSIONS FROM RESULTS OF TESTS.

The recoveries with the puddling method and the salt-fluorspar flux method, the latter on both dirty and washed chips, were practically identical and the same as the commercial recovery on these chips—that is, about 77 per cent on the weight of the chips and 87.5 per cent on the metallic content. As the use of salt-fluorspar flux had given better results on previous lots of chips in a gas furnace than the puddling method, it seems likely that the previous poorer results of the puddling method may have been largely due to the fact that the products of combustion in the gas furnace were continually sweeping over the metal as it was poked in. In run 97 there were no products of combustion, and the conditions were very like those in the iron-pot furnace.

It appears that, as puddling can not be done efficiently in a pit furnace, the use of salt-fluorspar flux is the most promising method of melting chips in pit furnaces. The temperatures required in the flux method are too high for the iron-pot furnace, and in that furnace the puddling method is preferable. In reverberatory melting the flux method is probably the better.

The choice of the three methods—puddling in the iron-pot furnace, use of the flux in pit or tilting crucible furnaces, or of the flux in a reverberatory furnace—for commercial refining can not be decided by

laboratory tests. Costs of fuel, flux, crucibles or iron pots, furnace repair, and labor, the output in regard to floor space and furnace cost, the yield of ingot, the quality of ingot, the ease of recovering buttons from slag or dross, and other factors must all be taken into account in making a choice. It seems likely that for very clean chips the iron-pot puddling process would be more economical, whereas for very dirty chips the flux methods, whether in crucibles or in a reverberatory furnace, would apparently have a slight advantage.

#### QUALITY OF INGOT FROM ALUMINUM CHIPS.

Ingot made from dirty aluminum borings or chips is necessarily somewhat less pure and less desirable than virgin metal. One would hesitate to use all boring ingot in important castings. However, in small castings, such as are molded on the bench, if they do not have to stand a pressure test, 10 per cent of good boring ingot can usually be added to 90 per cent of new metal and back gates without increase in the proportion of defective castings and without detriment to the quality of the castings. In the case of large castings of complicated design, it is a moot question among aluminum founders whether or not the use of boring ingot increases the amount of castings defective through cracking in the mold. In some cases which have come to the attention of the writers, boring ingot has given very low "hot shortness" tests, which would indicate a liability toward cracking in the mold. On the other hand, some foundry men have claimed that certain boring ingot, used in quantities even up to 30 per cent of the weight of the charge, markedly decreases the loss from cracking, although other ingot of practically the same composition markedly increases it. The matter is a difficult one to settle, as so many factors, such as the relative hardness or softness of the cores, the ramming of the mold, the temper of the sand, the design of the casting, the gating, and the temperature at which the metal is poured, may all have as much or more effect on the tendency to crack as has the composition of the metal. In fact, the analysis of the metal gives no real criterion as to its tendency to crack.

However, it is true that some metal does crack worse than other metal of the same composition. Some unpublished results of work by H. Goldberg have indicated that metal melted under conditions where waste gases from a strongly oxidizing flame may reach it is more likely to crack than would the same metal if melted under reducing conditions.

Although boring ingot does not command the price of new metal, a boring ingot not too high in impurities finds a ready market at a price not much below that of new metal, and this is the best indication that there are plenty of castings in which boring ingot may be satisfactorily used.



On the other hand, there has been some boring ingot on the market, evidently made from a miscellaneous mixture of aluminum, babbitt, and other white metal and brass and bronze chips, which resembles aluminum only in its color, and which is of very little value to the founder. For example, two samples of metal offered for sale by one smelting company were analyzed by the foundry buying them, with the following results:

*Composition of boring ingot made from various alloys.*

	Sample A.	Sample B.
Copper.....per cent..	17.4	15.3
Zinc.....do....	8.4	6.7
Iron.....do....	1.8	2.0
Silicon.....do....	2.4	2.8
Manganese.....do....	0.5	Trace.
Aluminum.....do....	69.5	73.2

Samples of boring ingot from another company, sold as G. M. B. (good merchantable brand) metal, not as any standard alloy, analyzed by the buyer, gave results as follows:

*Composition of G. M. B. metal.*

Sample No.	Copper.	Zinc.	Iron.	Silicon.	Lead.	Tin.	Aluminum.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
C.....	6.4	25.0	1.0	0.3	1.9	1.3	64.1
D.....	6.8	6.0	0.7	0.2	1.3	0.6	84.4
E.....	7.5	5.6	0.8	0.4	2.0	1.0	82.7

Another sample (F) from this firm, sold as No. 12 boring ingot, analyzed by the buyer, contained 7.2 per cent copper, 3.9 per cent zinc, 1.5 per cent iron, 0.4 per cent silicon, no lead nor tin, and 87 per cent aluminum.

Samples of boring ingot from a third company (all sold on the seller's analysis), melted in crucibles in coke fires with  $\text{NH}_4\text{Cl}$  flux, had the following compositions:

*Composition of boring ingot melted in crucible coke furnace with  $\text{NH}_4\text{Cl}$  flux.*

Sample No.	Copper.	Zinc.	Iron.	Silicon.	Manganese.	Aluminum.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
G.....	4.2	2.5	9.3	0.6	1.2	82.2
H.....	3.9	12.1	3.0	0.5	0.15	80.35
I.....	7.3	4.3	1.5	0.4	.....	86.5
J.....	7.7	0.3	2.8	1.9	.....	87.3
K.....	8.8	0.75	2.0	1.0	.....	87.45
L.....	9.4	1.0	1.3	0.3	.....	88.0
M.....	7.7	0.3	2.8	1.9	.....	87.3
N.....	9.7	0.7	1.1	1.0	.....	87.5



Three samples taken from various parts of a quantity of boring ingot offered for sale by a fourth company, and analyzed by the buyer, had the following composition:

*Composition of three samples of boring ingot.*

Sample No.	Copper.	Zinc.	Iron.	Silicon.	Aluminum.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
O.....	7.9	2.0	0.6	0.4	89.1
P.....	7.8	0.4	.7	.4	90.7
Q.....	7.9	2.0	.7	.4	89.0

The results of buyers' analyses of various lots of No. 12 boring ingot sold by a firm using the iron pot,  $\text{ZnCl}_2$ , puddling process, were as follows:

*Composition of No. 12 boring ingot.*

Sample No.	Copper.	Zinc.	Iron.	Silicon.	Aluminum.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
R.....	7.2	1.5	1.3	0.2	89.8
S.....	7.5	1.0	1.3	.3	89.9
T.....	7.6	1.5	1.5	.2	89.2
U.....	6.6	1.5	1.3	.2	90.4

Still another refining plant, using a reverberatory furnace and dealing with a mixture of No. 12 chips and emery grindings, has sold No. 12 boring ingot of the following composition, according to the buyer's analysis:

*Composition of ingot from No. 12 chips and emery grindings.*

Sample No.	Copper.	Zinc.	Iron.	Silicon.	Aluminum.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
V.....	7.3	2.4	1.3	0.2	88.8
W.....	8.6	0.4	1.1	.2	89.7
X.....	8.8	0.8	1.4	.4	88.6
Y.....	9.6	0.9	1.0	.4	88.1
Z.....	9.9	1.0	1.1	.3	87.7

The No. 12 alloy, made up from new metals, to contain 8 per cent copper, or purchased in market ingot made up from new metals, will analyze about 7.6 to 8.4 per cent copper, no zinc, 0.5 to 0.7 per cent iron, 0.25 to 0.4 per cent silicon, 91.65 to 90.5 per cent aluminum.

When melting in iron pots, about 0.2 per cent of iron is taken up from pots, ladles, and skimmers and about 0.05 per cent silicon from sand adhering to gates, making the average composition of the No. 12 castings sold 7.6 to 8.4 per cent copper, no zinc, 0.7 to 0.9 per cent iron, 0.30 to 0.45 per cent silicon, and 91.4 to 90.25 per cent aluminum. This alloy comprises 90 to 95 per cent of all cast aluminum made in the United States.

### EFFECT OF IMPURITIES.

The purity, or impurity, of boring ingot depends (1) on the purity of the castings from which the borings are made, (2) on the freedom of the borings from white metal, babbitt, or brass, (3) on their freedom from iron and on the efficiency of the magnetic separation, (4) on the presence or absence of dirt from which metals may be reduced, and (5) on the nature of the melting process and the temperature attained.

White metal, babbitt, brass, or bronze borings, if present, can not be efficiently separated from the aluminum borings and go into the melt. Copper, zinc, lead, tin, and antimony may come from these sources.

#### COPPER.

Of these metals copper is present not only in the No 12 alloy, but also in nearly all of the zinc-containing alloys, and its only bad effect would be to raise the copper content too high.

#### ZINC.

Zinc in No. 12 castings is frowned on by many purchasers, even if present only to the extent of a trace to less than one-half of 1 per cent, in some cases because it is known that zinc can not occur in No. 12 metal made wholly from virgin metal, but more often because a belief that the alloys of aluminum high in zinc give castings which may break in use under repeated shock is extended to the No. 12 alloy with but traces of zinc. This idea is prevalent in some sections of the country, whereas in other sections the zinc-containing alloys are much used. Alloys, such as the market "No. 31," containing about 3 per cent copper, 15 per cent zinc, 0.5 per cent manganese, 81.5 per cent aluminum, or a similar one containing 2 per cent copper, 19 per cent zinc, and 79 per cent aluminum, are stronger under tension or compression per unit weight than the No. 12 alloy, and, at normal prices for zinc, are cheaper per unit of weight or of volume. It does not appear to be satisfactorily proved that perfectly sound castings of these alloys are more likely to fail under repeated stress than the No. 12 alloy, as an alloy containing 19 per cent zinc has been satisfactorily used in one high-grade American automobile, and zinc-aluminum alloys, somewhat lower in zinc content, are used in European automobile plant practice almost to the exclusion of the No. 12 alloy, whereas the reverse is the case in the United States. It is claimed that because of the rougher roads in America the stresses a motor-car crank case, for example, has to endure are worse than in Europe, and that the zinc-containing aluminum alloys would not stand up as well as the No. 12 alloy under the more severe conditions in America. It is probable that the relative life of the crank cases

and other castings on the American and European motor cars and trucks in the European war will furnish some definite proof on this point.

It is true that it is more difficult to make complicated castings of zinc-containing alloys than of No. 12 alloy without having trouble from cracking in the mold, and that if castings with tiny cracks in them that have been undetected through lax inspection are put in service, failures are likely to result. But it is quite possible to make distinctly complicated castings of the zinc-containing alloys without defects if proper foundry practice is used.

Rosenhain and Archbutt <sup>a</sup> have made endurance tests of various aluminum alloys and do not find that the zinc-containing alloys are liable to fail under repeated shock. They indeed advocate the use of an alloy consisting of 20 per cent zinc and 80 per cent aluminum, or one containing 3 per cent copper, 25 per cent zinc, and 72 per cent aluminum, rather than those lower in zinc content, on account of the greater strength per unit weight.

A mixture of aluminum borings from various sources in the United States will consist mostly of No. 12 alloy, but may also contain some borings from alloys containing 10 to 30 per cent zinc, consequently the ingot from such mixed borings may contain zinc in varying amounts, as is shown by the analyses previously given.

As zinc volatilizes quite readily from aluminum alloys at the temperatures necessary in melting chips, some of the zinc content of alloys high in zinc will be lost in melting. A few hundredths of a per cent of zinc in No. 12 alloy might come from the taking up of zinc from  $\text{ZnCl}_2$  used as a flux, whereas a few tenths of a per cent or more of zinc indicates that some scrap, or else No. 12 ingot made in part from scrap, has been used. There is no satisfactory evidence to show that a zinc content up to 0.5 per cent at least in No. 12 alloy is any detriment to the strength or endurance of the castings. As the presence of zinc indicates the presence of scrap and presumably of boring ingot which may, under conditions not clearly understood, give rise to the cracking of castings in the mold or to other foundry defects, some progressive founders are using virgin pure aluminum and copper and preparing their own No. 12 alloy instead of purchasing the prepared alloy, and are using analyzed boring ingot in varying proportions, or leaving it out altogether, according to the liability of the individual casting that is to be poured from the metal to the troubles which boring ingot sometimes causes. With such rational utilization of boring ingot it finds a legitimate use in the foundry.

---

<sup>a</sup> Rosenhain, W., and Archbutt, S. L., Alloys of aluminum and zinc: Engineering, vol. 93, 1912, p. 578.



## LEAD.

Lead is not a desirable constituent in aluminum alloys, as it does not alloy with aluminum, but is merely inclosed in small globules. The lead, on account of its high specific gravity, will tend to collect at the bottom of the casting, and any appreciable amount tends to weaken the alloy. Such boring ingot as that shown by analyses C, D, and E is of poor quality on account of the lead content.

## TIN.

Tin is to be classed as a less harmful impurity. It tends to soften the alloy without noticeably affecting the strength. A small producer of high-grade automobiles has used an alloy containing 5 per cent copper and 3 per cent tin without any apparent disadvantage save a slightly lower strength. Such an alloy made from new metals has no obvious advantage to counterbalance the high cost of the tin, but a few tenths of a per cent of tin is not to be feared in boring ingot, save that, as it usually comes from admixed solder or from bronze chips, it may be accompanied by lead.

## ANTIMONY.

The effect of antimony on aluminum alloys is not well known. Its occurrence in boring ingot would indicate admixture of babbitt with the chips.

## MANGANESE.

Manganese in appreciable proportion would indicate that the ingot was made from material containing more or less hard sheet clippings. In small amount, with a high zinc content, for example, in analysis H, it would indicate that the mixture melted contained some No. 31 alloy chips.

In percentages up to say 0.2 per cent, manganese is not a detriment to No. 12 alloy, and some founders think that 0.1 or 0.2 per cent is distinctly desirable, although manganese is seldom intentionally added to No. 12 alloy, its source being some form of scrap from hard-sheet aluminum.

## IRON.

Iron is the main impurity in boring ingot made from No. 12 chips and comes from the admixture in the machine shop of iron or steel chips with the aluminum chips. Commercial magnetic separation of iron from such chips may leave, especially in the case of very oily chips, 0.5 to 1.0 per cent of free iron in the mass. All of this free iron goes into solution when the mass is melted. Save for the free iron, no increase in iron content would be shown in melting chips in crucibles and not over 0.25 per cent increase in melting in iron pots by the puddling process.



## INTERRELATED EFFECTS OF IRON, COPPER, AND ZINC.

As high as 2 per cent of iron in the No. 12 alloy would probably give too brittle an alloy, but a content of between 1 and 1.5 per cent does not appear to have a notably detrimental effect. If the iron content rises above 1 per cent, the copper content should be correspondingly reduced below 8 per cent in order to prevent brittleness, as iron acts much like copper in hardening and strengthening aluminum.

Cast No. 12 alloy, unannealed, consists of a needle-like  $\text{CuAl}_2$ -Al eutectic embedded in the groundmass of solid solution of copper in aluminum. Iron in aluminum causes a similar needle-like structure in a groundmass of solid solution, and as the properties of the No. 12 alloy depend largely on the proportions of needles and groundmass too high an iron content will give too much of the needle-like material. If zinc be added to the No. 12 alloy, more of the needle-like eutectic is thrown out than without it, so that in the commercially useful ternary alloys of copper, aluminum, and zinc the copper is used in smaller amount as the zinc is used in larger amount. For example, an alloy containing 7.5 per cent copper and 1 per cent zinc has about the same tensile strength and elongation as one containing 8 per cent copper. With 20 per cent zinc not over 4 per cent copper and with 30 per cent zinc not over 2 per cent copper should be used, else too brittle an alloy will result.

By proper reduction in the copper content the effect of high iron content and of a little zinc, introduced by the use of some boring ingot, can largely be neutralized, and the resulting alloy will still have very closely the properties of No. 12 alloy as regards foundry behavior, shrinkage, and tensile strength and elongation.

## SILICON.

Silicon is generally considered to be an undesirable impurity and, especially if in the form of graphitic silicon, to be conducive to brittleness. It is always present in small amount in commercial aluminum, owing to the difficulty of completely freeing from silica the bauxite used in its manufacture.

In remelting gates to which molding sand adheres, or in melting chips that contain siliceous dirt some silica may be reduced to silicon which alloys with the aluminum and an equivalent amount of aluminum is oxidized and lost.

This reduction of silica by aluminum is not marked below  $800^{\circ}\text{C}$ . and even when melting in fire clay graphite crucibles only a very small content of silicon is taken up if the temperature is kept low. As the temperature increases reduction takes place more readily, so that at the temperatures necessary in melting chips some slight in-

crease in silicon content is to be expected if they contain siliceous dirt. The use of a flux that will dissolve or soak up the dirt and so take it out of contact with the metal might be expected to decrease the amount of silicon taken up at any given temperature.

The results of analyses of commercial lots of boring ingot as shown on pages 73 and 74 give no information as to the relative purity of ingot made from the same lot of chips by different methods of melting, as any such differences are doubtless overbalanced by variations in the purity of the alloys from which the chips were taken and by variations in the content of free iron and dirt.

#### COMPOSITION OF INGOT FROM ELECTRIC FURNACE.

Analyses were made by the writers of the ingot obtained in the 15-pound melts of chips from lot B in the electric crucible furnace, in run 69 when a yield of 58 per cent on the original weight of the borings was obtained with sal ammoniac flux, in run 70 when a yield of 71.5 per cent on the original weight was obtained with NaCl-CaF<sub>2</sub> flux, and in run 83 when a yield of 89 per cent was obtained with the NaCl-CaF<sub>2</sub> flux on the fraction over 14-mesh in size. The results of the analyses were as follows:

*Composition of ingot from melts in electric furnace.*

Run No.	Copper.	Iron.	Silicon.	Manganese.	Zinc.	Aluminum (by difference).
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
69	7.40	1.87	0.34	Trace.	None.	90.39
70	7.39	1.92	.36	Trace.	None.	90.33
83	7.42	1.25	.34	Trace.	None.	90.99

The only appreciable difference is in the lower iron content of the fraction over 14-mesh, which indicates that most of the free iron left in this sample after commercial magnetic separation was under 14-mesh, an assumption which is corroborated by visual examination. Notwithstanding the difference of 13.5 per cent in the yields of runs 69 and 70, the composition of the ingot obtained is identical. As the run with the salt and fluorspar flux shows no appreciable increase in silicon content although the borings contained over 12 per cent of siliceous dirt, and as, unless the borings are melted in an iron pot, the iron content can not be greater than the original content in the alloy from which the chips were made plus the free iron not removed by the magnetic separator, the criticism that the high temperature needed to make this flux fluid results in ingot higher in impurities than by other methods of melting does not seem well founded.

**NECESSITY FOR CARE AND CLEANLINESS IN COLLECTING CHIPS.**

The high loss in melting chips caused by difficult coalescence and subsequent oxidation of globules entangled in the dross, and the poor quality of boring ingot are due primarily to lack of care and cleanliness in the collection of the chips in the machine shop. If the aluminum chips are kept free from brass, bronze, solder, white-metal, or iron chips, and also free from dust and floor sweepings, the only foreign material in them will be the cutting oil or cutting lubricant. By promptly centrifuging the chips these lubricants can be removed and recovered, and the chips left in a better condition for melting. If chips are thus collected and centrifuged and then stored in covered receptacles so that dust can not settle on them, clean chips will result.

The value of clean chips is so much greater than that of dirty chips, mixed with iron chips, that the slight increase in cost of careful collection over that of collecting by slipshod methods would be many times repaid and on the large amount of chips produced by even a comparatively small manufacturer of motor cars the saving would be well worth while.

As at least two methods of melting chips now in use by refiners, the puddling process and the  $\text{NaCl-CaF}_2$  process, will give a recovery of 90 per cent or higher on absolutely clean chips and produce boring ingot of good quality, the responsibility for the waste of aluminum in melting chips, which is large in the aggregate, rests at the door of the machine shop that permits iron and dirt to get into the chips, rather than at the door of the foundry or refining plant that attempts to turn the chips into usable ingot.

Foundry refuse, such as buttons recovered from skimmings can not, in the nature of the material, be collected free from foreign matter. Hence such material, as well as carelessly collected chips, offers difficulties in melting owing to reluctance of the globules formed in melting to coalesce.

**PROBABLE RELATIVE USEFULNESS OF DIFFERENT MELTING METHODS.**

Of the methods suggested for melting aluminum chips, some are of doubtful practicability. Feeding the chips into the molten electrolytic bath used for producing pure aluminum is plainly useless in the case of dirty chips, on account of fouling the rather expensive electrolyte. The method is not well understood by founders and refiners, as the details of the electrolytic process are kept secret by the producers of aluminum, so that even on clean chips the method would be of use only to those producers.



Briquetting involves the use of expensive machinery, and although the use of briquets might slightly decrease the melting loss, would probably materially decrease the melting time and fuel cost, and would certainly increase the ease of handling, yet it is likely that a briquetting machine of the type necessary to produce solid briquets would have to be kept in constant use in order that the interest on the first cost should not eat up the savings.

Hence briquetting would only be practiced by large refiners who have large amounts of aluminum chips to handle or have other uses for such apparatus. Even at such plants the question whether briquetting would pay is not yet fully answerable, although the process deserves serious consideration.

The use of electric furnaces, vacuum furnaces, retorting furnaces, or other devices which merely prevent the access of air without providing for the promotion of coalescence by stirring seems to offer little promise.

The method so often advocated of pushing the borings beneath a heel of molten metal seems to owe any usefulness it may have to the pushing and not to the heel. It is an approach to the puddling process, which seems to be the best modification of the scheme.

In the iron-pot puddling process, which is evidently of much value on clean borings, the vital points are to keep the temperature very low, practically at the melting point, to prevent oxidation, and constantly stir the pasty mass to promote coalescence. The process requires constant hand labor, does not seem as useful on very dirty borings, and could hardly be applied to large units.

Possibly some mechanically stirred furnace, or one in which pressure was constantly applied, analogous to the Côte and Pierron suggestion or to the Montefiore method for melting blue powder, could be devised that would have the advantages of the puddling process without the high labor cost, but no such apparatus had yet been put forth, and the making of one that would have a low upkeep cost presents serious difficulties.

The use of volatile fluxes such as sal ammoniac and zinc chloride is of advantage at the end of the puddling process, but melting with these fluxes without constant stirring gives very low recoveries.

The use of fluxes that form molten covers seems to be of real advantage only when rather large quantities of the flux are used, and the promotion of coalescence by the flux soaking up dirt and oxide seems to be of more importance than the prevention of oxidation by exclusion of air. Although fluxes melting at or below the melting point of aluminum are theoretically preferable to those of higher melting point, it is difficult to find such low-melting fluxes chemically inert to aluminum without using mixtures containing materials such as

lithium chloride or potassium chloride, the first of which is very costly and the second relatively so, or hygroscopic materials such as calcium or magnesium chlorides. Common salt, the cheapest flux of the molten-cover type, seemingly gives the best results, when the cost and convenience of the flux is considered. Still better are fluxes that have a real solvent action on aluminum oxide and on dirt in general, such as the mixtures of alkali chlorides and fluorides. With such fluxes better absolute recoveries could probably be obtained by using very low melting mixtures, but as large amounts of these fluxes are necessary their cost is prohibitive, and the cheap flux consisting of 85 parts common salt and 15 parts fluorspar, used in large amounts, 20 to 30 per cent of the weight of the chips, mixed with the chips before charging, and the whole heated in crucibles till the flux is fairly fluid, seems the most useful. On clean borings it would probably not give higher results than the iron-pot puddling process, but it may be preferable even then as constant stirring is not required, and thus the labor cost would be less, and because it might make large-scale reverberatory melting feasible. The experiments on reverberatory melting indicated this possibility but were on too small a scale to be conclusive. If chips must be melted in pit fires, the flux method is the most suitable; if iron-pot furnaces are to be used, the puddling process is the better.

There is also a possibility that dirty borings washed with dilute caustic soda, subjected to wet concentration, as on Wilfley tables, and then dried by centrifuging, would yield a clean product on which the melting loss would be low. Lack of suitable apparatus made the actual experiments inconclusive, but the scheme seems worthy of a trial on a commercial scale.

However, if the machine shops producing aluminum chips would keep the borings clean and free from contamination, it would not be necessary to go to the expense of removing foreign matter.

#### ACKNOWLEDGMENTS.

Grateful acknowledgment is due to the department of chemistry of Cornell University for the use of its laboratories, the facilities of which have been extended to the Bureau of Mines under a cooperative arrangement; to Prof. Bancroft, of Cornell, whose advice, particularly in regard to the theory of coalescence as the basis for a successful method of melting chips, has been freely drawn on, and under whose supervision the aid of the junior author, Mr. G. M. James, of Cornell, was extended in making many of the tests; to Dr. J. M. Lohr and Mr. E. L. Mack, of the Bureau of Mines, for aid in experimental work; and to Messrs. C. B. Bohn, A. B. Norton, W. R. Laird, J. Sillman, A. J. Hall, H. B. Swan, W. H. Goldstine,

G. H. Clamer, M. T. Hearley, F. J. Root, C. H. Jumper, and H. P. Cowen, as well as to another who does not wish his name to appear, for material supplied for the investigation or for information.

### PUBLICATIONS ON MINERAL TECHNOLOGY.

A limited supply of the following publications of the Bureau of Mines is temporarily available for free distribution. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

BULLETIN 16. The uses of peat for fuel and other purposes, by C. A. Davis. 1911. 214 pp., 1 pl., 1 fig.

BULLETIN 42. The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs.

BULLETIN 45. Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 47. Notes on mineral wastes, by C. L. Parsons. 1912. 44 pp.

BULLETIN 64. The titaniferous iron ores of the United States, their composition and economic value, by J. T. Singewald, jr. 1913. 145 pp., 16 pls., 3 figs.

BULLETIN 81. The smelting of copper ores in the electric furnace, by D. A. Lyon and R. M. Keeney. 1915. 80 pp., 6 figs.

BULLETIN 84. Metallurgical smoke, by C. H. Fulton. 1915. 94 pp., 6 pls., 15 figs.

BULLETIN 85. Analyses of mine and car samples of coal collected in the fiscal years 1911 to 1913, by A. C. Fieldner, H. I. Smith, A. H. Fay, and Samuel Sanford. 1914. 444 pp., 2 figs.

BULLETIN 92. Feldspars of the New England and Northern Appalachian States, by A. S. Watts. 1915. 181 pp., 8 pls., 22 figs.

BULLETIN 100. Manufacture and uses of alloy steels, by H. D. Hibbard. 1915. 78 pp.

BULLETIN 104. Extraction and recovery of radium, uranium, and vanadium from carnotite, by C. L. Parsons, R. B. Moore, S. C. Lind, and O. C. Schaefer. 1915. 124 pp., 14 pls., 9 figs.

BULLETIN 106. The technology of marble quarrying, by Oliver Bowles. 1916. 174 pp., 12 pls., 33 figs.

TECHNICAL PAPER 8. Methods of analyzing coal and coke, by F. M. Stanton and A. C. Fieldner. 1913. 42 pp., 12 figs.

TECHNICAL PAPER 14. Apparatus for gas-analysis laboratories at coal mines, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 7 figs.

TECHNICAL PAPER 32. The cementing process of excluding water from oil wells, as practiced in California, by Ralph Arnold and V. R. Garfias. 1912. 11 pp., 1 fig.

TECHNICAL PAPER 38. Wastes in the production and utilization of natural gas and means for their prevention, by Ralph Arnold and F. G. Clapp. 1913. 29 pp.

TECHNICAL PAPER 39. The inflammable gases in mine air, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 2 figs.

TECHNICAL PAPER 43. The influence of inert gases on inflammable gaseous mixtures, by J. K. Clement. 1913. 24 pp., 1 pl., 8 figs.

TECHNICAL PAPER 50. Metallurgical coke, by A. W. Belden. 1913. 48 pp., 1 pl., 23 figs.

TECHNICAL PAPER 66. Mud-laden fluid applied to well drilling, by J. A. Pollard and A. G. Heggem. 1914. 21 pp., 12 figs.



TECHNICAL PAPER 68. Drilling wells in Oklahoma by the mud-laden fluid method, by A. G. Heggem and J. A. Pollard. 1914. 27 pp., 5 figs.

TECHNICAL PAPER 70. Methods of oil recovery in California, by Ralph Arnold and V. R. Garfias. 1914. 57 pp., 7 figs.

TECHNICAL PAPER 76. Notes on the sampling and analysis of coal, by A. C. Fieldner. 1914. 59 pp., 6 figs.

TECHNICAL PAPER 88. The radium-uranium ratio in carnotites, by S. C. Lind and C. P. Whittemore. 1915. 29 pp., 1 pl., 4 figs.

TECHNICAL PAPER 95. Mining and milling of lead and zinc ores in the Wisconsin district, Wisconsin, by C. A. Wright. 1915. 39 pp., 2 pls., 5 figs.

TECHNICAL PAPER 110. Monazite, thorium, and mesothorium, by K. L. Kithil. 1915. 32 pp., 1 fig.

TECHNICAL PAPER 111. Safety in stone quarrying, by Oliver Bowles. 1915. 48 pp., 5 pls., 4 figs.

TECHNICAL PAPER 126. The casting of clay wares, by T. C. McDougal. 1916. 26 pp., 6 figs.

TECHNICAL PAPER 128. Quarry accidents in the United States during the calendar year 1914, compiled by A. H. Fay. 1915. 45 pp.

# INDEX.

## A.

	Page.
Air, absence of, effect on melting aluminum scrap.....	37, 38
Alloys, aluminum, losses of.....	10
recovery of.....	9, 10
Aluminum, coalescence of.....	22, 23, 26
consumption in United States.....	9
fusion temperature of.....	67
in boring ingot, proportions of.....	73, 74
melting of, electric power required for.....	67
nitridation of.....	20, 21
oxidation of.....	20
<i>See also</i> Scrap aluminum.	
Aluminum carbide, formation of.....	22
Aluminum chloride, reduction of.....	20
use as flux.....	14
results of.....	34, 48
Aluminum oxide, reduction of.....	20
Aluminum powder, uses of.....	20
Alundum-magnesite lining for furnace, tests with.....	66-69
Ammonium chloride, use as flux.....	16
results of.....	34, 48
<i>See also</i> Sal-ammoniac.	
Anderson, J., on fluxes for melting aluminum scrap.....	19
Antimony in boring ingot, effect of.....	77
Archbutt, S. L., on zinc in aluminum alloys..	76
Arndt, K., on fluxes for melting aluminum scrap.....	19

## B.

Bancroft, W. D., acknowledgments to.....	82
cited.....	60
Bennett, H. S., experiments of.....	21
Bleeker, W. F., on melting of blue powder.....	24, 25
"Blue powder," analysis of.....	24
coalescence of.....	24, 26
definition of.....	24
kinds of.....	24
melting of.....	25, 26
Bohn, C. B., acknowledgments to.....	82
Borax, melting point of.....	16
use as flux.....	14, 15, 45, 70
Boric acid, use in flux.....	16
Boring ingot. <i>See</i> Ingot.	
Borings, briquetting of, advantages of.....	19
contamination of.....	6, 7
dirty, results of tests with.....	44
melting of.....	6, 27
losses in.....	10
price of.....	8
sized, melting of, results of.....	58, 59
sizing of, effect of.....	58
washed, results of melting.....	61-63
<i>See also</i> Chips; Scrap aluminum.	

## Page.

Boron fluoride, use in flux.....	15
Briquetted chips, results of melting.....	48-51
Briquetting of chips, advantages of.....	19, 26, 81
Bureau of Mines, investigations of.....	5
Burgess, G. K., on coalescence of tin.....	23
on melting point of flux.....	17

## C.

Calcium chloride, use as flux.....	13, 14, 43, 44
Calcium fluoride, use in flux.....	15
Carnallite, melting point of.....	16
use as flux, results of.....	42
Carpenter, H. C. H., on nitridation of aluminum.....	21
Castings, aluminum, in motor-car construction, amount of.....	8
machining off, losses from.....	8, 9
melting of.....	6
Caustic-soda solution, washing of chips with.....	63-65, 82
Centrifuge for drying borings, merits of.....	29, 30
Chips, collection of, need of care in.....	80
contamination of.....	6, 7
disposal of.....	7
melting of, methods for.....	11-26
size of.....	7
effect on recovery.....	39, 40
sized, recovery from melting.....	65
uses for.....	10
washing of, results of.....	60-65
Chlorine, melting of chips in presence of.....	35
Chubb, L. W., on coalescence of aluminum..	23
Clamer, G. H., acknowledgments to.....	82
Clippings, sheet, melting loss of.....	5, 6
Copper, fusion temperature of.....	68
in boring ingot, effect of.....	75, 78
proportion of.....	73, 74, 79
Cornell University, acknowledgments to.....	82
Cote, E. F., on melting of blue powder.....	25
Coulson, J., on melting aluminum scrap.....	12
Cover, molten. <i>See</i> Fluxes.	
Cowen, H. P., acknowledgments to.....	82
Crucibles, melting of borings in.....	27, 30, 31
disadvantages of.....	29
relative merits of.....	71
results of.....	70
Cryolite, melting point of.....	17
use as flux.....	12, 15, 17
Cutting compound, water-soluble, washing of chips with.....	60

## D.

De Ville, —, puddling method used by.....	13
Desch, C. H., on fluxes for melting aluminum scrap.....	16

	Page.		Page.
Dirt on borings, effect of.....	39, 40, 44	"Heel" of metal, melting of briquet in.....	49
Dross, effect of air on.....	39	melting chips in, relative value of.....	81
effect of water on.....	39	results of.....	38, 62
from melting of aluminum, formation of.....	22	Hirsch, E. F., on briquetting borings.....	19
remelting of.....	30, 31	Hydrochloric acid gas, melting of chips in	
washing of.....	30	presence of, results of.....	35
E.		I.	
Echevarri, J. T. W., on melting aluminum		Ilkinsky, W., on fluxes for melting aluminum	
scrap.....	12	scrap.....	18
on oxidation of aluminum.....	20	Indirect arc furnace, description of.....	66
Edwards, C. A., on nitridation of aluminum.....	21	melting chips in, difficulties in.....	68
Electric furnace, composition of ingot from.....	79	tests with.....	66-69
for melting aluminum scrap.....	12	Ingalls, W. R., cited.....	24, 25
difficulties with.....	68, 69	on analysis of blue powder.....	24
relative efficiency of.....	68, 81	Ingot from borings, composition of.....	73, 74, 79
tests with.....	55, 56	price of.....	8
<i>See also</i> Indirect arc furnace; Resistor		purity of, factors governing.....	72, 75
furnace.		uses for.....	72
Electrolyte for production of aluminum, com-		Iron, calorizing of.....	23
position of.....	17, 18	contamination of chips by.....	7
England, melting of aluminum in.....	27	in boring ingot, effect of.....	77
F.		proportions of.....	73, 74, 79
Fedotieff, P. P., on fluxes for melting alumi-		Iron pots, melting of aluminum in.....	28, 29, 56, 57
num scrap.....	18	advantages of.....	29, 71
Fichter, F., on nitridation of aluminum.....	21	recovery from.....	29
Fluoride fluxes, results of tests with.....	45, 46, 48	J.	
Fluorspar, use as flux.....	16, 18, 31, 46, 47	James, G. M., acknowledgments to.....	82
Flux, commercial, composition of.....	41	Johnson, W. McA., cited.....	24
method of using.....	41	Jones, J. L., on use of fluxes.....	14, 45
results of tests with.....	41	Jumper, C. H., acknowledgments to.....	82
for melting aluminum scrap.....	13-19, 26	K.	
kinds of.....	81, 82	Keeney, R. M., cited.....	24
melting points of.....	16-18	L.	
merits of.....	40, 81	Laird, W. R., acknowledgments to.....	82
Foundry wastes, contamination of.....	7	Lane, H. M., on use of zinc chloride as flux..	13
recovery from.....	8	Le Chatelier, H., on disintegration of calcium	
<i>See also</i> Borings; Chips; Scrap aluminum.		aluminum alloys.....	13
Furnace for melting chips, description of.....	28	Lead, fusion temperature of.....	68
<i>See also</i> Electric furnace; Resistor furnace;		in boring ingot, effect of.....	77
Retort furnace; Reverberatory furnace;		proportions of.....	73
Vacuum furnace.		Lithium chloride, use in flux.....	15
Furnace linings, tests of.....	66-69	Loewenstein, W., on fluxes for melting alumi-	
G.		num scrap.....	19
Gasoline, washing of chips with.....	61, 62	Lohr, J. M., acknowledgments to.....	82
Gleason, E. D., on use of fluxes for melting		Lorenz, R., on electrolyte for production of	
aluminum.....	45	aluminum.....	18
Goldberg, H., cited.....	72	Lyon, D. A., cited.....	24
Goldstine, W. H., acknowledgments to.....	82	M.	
Graphite crucible lining for furnace, test with.	69	Mack, E. L., acknowledgments to.....	82
Grindings, aluminum, melting of, recovery		Magnesium, use as flux.....	12, 14
from.....	31	Manganese in boring ingot, effect of.....	77
emery, in boring ingots, effect of.....	74	proportions of.....	73, 79
Guertler, W., on fluxes for melting aluminum		Matignon, C., on nitridation of aluminum.....	20
scrap.....	16	on reduction of magnesia by aluminum..	14
H.		Mellen, G., on melting of aluminum scrap... 15, 16	
Hall, A. J., acknowledgments to.....	82	Mellen, J. G., cited.....	14
Hall, C. M., on fluxes for melting aluminum		Mellen, W. F., cited.....	14
scrap.....	17	Mercury, coalescence of.....	23
Hansen, C. A., cited.....	24	Merica, P. D., on coalescence of tin.....	23
"Hard sheet" aluminum, melting of.....	6	Micks, R., on use of zinc chloride as flux....	13, 14
Hearley, M. T., acknowledgments to.....	82		



	Page.		Page.
Mierzinski, S., on use of fluorspar as flux.....	18	Salt, use as flux.....	15, 16, 30, 31
Moldenhauer, W., on fluxes for melting aluminum scrap.....	19	relative value of.....	82
Morrison, W. M., on melting of aluminum scrap.....	15, 23	results of.....	42-44, 66
N.		Salt-fluorspar flux, relative efficiency of.....	57, 71
Neumann, B., on electrolyte for production of aluminum.....	18	tests with.....	46-48, 52, 53, 55-59, 62, 63, 66, 67, 69-71
Nitrogen in aluminum, effect of.....	21	results of.....	54
proportion of.....	21	Salt-peter, as flux.....	16
removal of.....	21	Schoop, M. V., cited.....	45
Norton, A. B., acknowledgments to.....	82	on coalescence of aluminum globules.....	23
O.		on fluxes used in melting aluminum scrap.....	14, 15
Oil on borings, effect of.....	39, 40	Schultz, F., on flux for melting aluminum alloys.....	16
Olsen, H., on electrolyte for production of aluminum.....	18	Scrap aluminum, losses in melting, experiments regarding.....	5
Oxygen, attraction of aluminum for.....	20	methods of melting.....	11
P.		production of.....	5
Pannell, E. V., on composition of fluxes.....	15	<i>See also Borings; Castings; Chips.</i>	
Pascal, P., on electrolyte for production of aluminum.....	17	Scrap cable, melting of.....	6
Peacock, —, on production of carbonitride.....	22	Scrap wire, melting of.....	6
Peterson, P. E., on blue powder.....	24, 25	Seligman, R., on melting aluminum chips.....	15, 27
Pierron, P. R., on melting of blue powder.....	25	on coalescence of aluminum globules.....	23
Plato, W., cited.....	18, 45	Serpek, —, production of aluminum nitride by.....	22
Potassium bisulphate, use in flux.....	15	Silicon in boring ingot, effect of.....	78, 79
Potassium chloride, use in flux.....	15, 16	proportions of.....	73, 74, 79
Potassium fluoride, use in flux.....	15	reduction of.....	78
Potassium nitrate, use as flux.....	16	Sillman, J., acknowledgments to.....	82
<i>See also Salt-peter.</i>		Skimmings, melting of, results of.....	55
Potassium sulphate, use in flux.....	15	recovery of aluminum from.....	11
Puddling, continuous, on melting aluminum, results of.....	36, 48	Skinner, C. E., on coalescence of aluminum globules.....	23
Puddling process for melting chips, relative merits of.....	56, 57, 71, 81	Sodium bisulphate, use in flux.....	15
Punchings, sheet, melting loss of.....	5, 6	Sodium chloride. <i>See Salt.</i>	
R.		Sodium fluoride, use in flux.....	14, 15
Reducing atmosphere, effect on melting aluminum scrap.....	37	Sperry, E. S., on briquetting borings.....	19
Resistor furnace, tests with.....	70, 71	on melting aluminum scrap.....	11
Retort furnace, for melting of aluminum.....	26	on nitridation of aluminum.....	21
relative value of.....	81	Steel, calorizing of.....	23
Retorting of chips, results of.....	36, 48	Sulphur, use in flux.....	16
Reverberatory furnace, experimental, details of.....	51	Swan, H. B., acknowledgments to.....	82
tests in.....	52, 53	● T.	
melting of aluminum in.....	12, 30, 32, 51-54	Tchijerski, N. P., on nitrogen in aluminum.....	21
relative merits of.....	71	Temperature in tests, variations of.....	33, 34
Richards, J. W., cited.....	68	Thaulow, E., on fluxes used in melting aluminum scrap.....	15
on electrolytic refining of zinc.....	25	Thermit reaction, definition of.....	20
on nitridation of aluminum.....	22	Thierry, C. V., on melting blue powder.....	26
on use of fluxes.....	16	Tin, coalescence of.....	23
Roerber, E. F., on blue powder.....	24	fusion temperature of.....	68
Root, F. J., acknowledgments to.....	82	in boring ingot, effect of.....	77
Rosenhain, W., on zinc in aluminum alloys.....	76	proportion of.....	73
Rossi, A. J., on effect of titanium on aluminum.....	21	Titanium, removal of nitrogen in aluminum by.....	21
S.		Trood, S., on zinc dust for sherardizing.....	24
Salammoniac, as flux.....	16, 27	U.	
tests with.....	53, 56	United States, aluminum losses in.....	9
results of.....	44, 54	consumption of aluminum in.....	9
<i>See also Ammonium chloride.</i>		United States Geological Survey, on recovery from aluminum chips.....	9
		V.	
		Vacuum furnace, for melting aluminum.....	15, 26
		relative value of.....	81
		Vickers, C., on melting aluminum scrap.....	11
		on nitridation of aluminum.....	21

W.		Page.		Page.
Water, washing of chips with.....		60	Zavelberg, A., on smelting of zinc.....	16
Weber, H., on use of fluxes.....		14	Zinc, fusion temperature of.....	68
West, T., cited.....		24	in boring ingot, effect of.....	75, 76, 78
Wright, J., cited.....		45	proportion of.....	73, 74, 79
on electrolyte for production of aluminum		17	Zinc chloride as flux.....	11, 13, 14, 16, 29, 30
Z.			disadvantages of.....	27
Zappi, E. V. on effect of fine division o			relative merits of.....	35
aluminum.....		20	tests with.....	53, 58, 59
			results of.....	34, 42, 48, 54



DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

OPERATING DETAILS OF GAS PRODUCERS

BY

R. H. FERNALD



WASHINGTON  
GOVERNMENT PRINTING OFFICE

1916



The Bureau of Mines, in carrying out one of the provisions of its organic act—to disseminate information concerning investigations made—prints a limited free edition of each of its publications.

When this edition is exhausted, copies may be obtained at cost price only through the Superintendent of Documents, Government Printing Office, Washington, D. C.

The Superintendent of Documents *is not an official of the Bureau of Mines*. His is an entirely separate office, and he should be addressed:

SUPERINTENDENT OF DOCUMENTS,  
*Government Printing Office,*  
*Washington, D. C.*

The general law under which publications are distributed prohibits the giving of more than one copy of a publication to one person. The price of this publication is 10 cents.

*First edition. August, 1916.*

# CONTENTS.

	Page.
Introduction.....	5
Present status of producer-gas plants.....	5
Producer-gas plants in Europe.....	7
Slagging gas producer.....	9
Producer-gas plant compared with steam turbine.....	10
Character of fuel used in gas producers.....	11
Tests of low-grade fuel.....	12
Data on fuel used at commercial plants.....	14
Pounds of fuel per square foot of fuel-bed area per hour.....	19
Fuel consumption of up-draft plants using anthracite coal.....	20
Fuel consumption of up-draft plants using bituminous coal.....	21
Fuel consumption of up-draft plants using lignite.....	22
Fuel consumption of up-draft plants using wood.....	22
Fuel consumption of down-draft plants using bituminous coal.....	23
Fuel consumption in down-draft gas producer using lignite.....	23
Fuel consumption of double-zone plants using bituminous coal.....	23
Conclusions regarding coal consumption.....	24
Pounds of fuel per horsepower-hour.....	24
Depth of fuel bed.....	27
Composition of gas.....	28
Heat value of gas.....	30
Cubic feet of gas per pound of fuel.....	31
Time between fuel charges and weight of fuel charges.....	32
Time between pokings of fuel bed.....	33
Trouble from holes and channels in fuel.....	34
Methods of overcoming variations in quality of gas.....	35
Methods of cleaning gas generators.....	36
Serious cleaning troubles.....	38
Time between periods of drawing producer fires.....	39
Methods of scrubbing and cleaning gas.....	40
Time between renewals of scrubber material.....	41
Opinions regarding scrubbing and cleaning results.....	42
Manner in which tar gives trouble.....	43
Trouble caused by lampblack.....	45
Relation of tar or lampblack in engine to cost of lubrication.....	46
Burner trouble caused by tar and lampblack.....	47
Trouble from sulphur in producer gas.....	47
Proportion of sulphur allowable in fuel.....	48
Auxiliary power used.....	48
Trouble from producer auxiliaries.....	50
Time required to bring gas producers to full gas production after stand-by.....	51
Stand-by fuel.....	52
Quantity of water used by producer plants.....	53
Quantity of vaporizer water used.....	53
Quantity of scrubber water used.....	54
Quantity of water required to cool engine.....	55

	Page.
Lubricants used.....	56
Preignitions and their causes.....	57
Changing or cleaning igniters.....	59
Gas pressures and distances for gas delivery.....	60
Recovery of by-products from waste liquors from scrubbers, washers, etc.....	60
Commercial use made of tar produced.....	60
Commercial utilization of carbon residue from oil-gas producers.....	61
Enrichment of producer gas.....	61
Temperature required at furnace.....	61
Spare capacity and power actually available.....	61
Uses of producer-gas plants.....	62
Response of producer to sudden changes in demand.....	63
Period of continuous operation.....	64
Reliability and length of service of plants.....	66
Publications on fuel technology.....	69



# OPERATING DETAILS OF PRODUCER-GAS INSTALLATIONS.

---

By R. H. FERNALD.

---

## INTRODUCTION.

In 1900, as far as available records show, there were only two producer-gas power installations in the United States.

In June, 1915, the number probably exceeded 1,000. Of this number, some 84.5 per cent comprised small plants using anthracite, and only about 15 per cent of the total number utilized bituminous coal and lignite. In addition to these, two plants used wood, and three or four used oil.

In the main, however, the larger sized units are operating on bituminous coal and lignite.

During the early years of these plants little success was anticipated from the use of bituminous coals and lignites, but in 1904 and 1905 it was fully demonstrated at the Government testing station at St. Louis that these fuels could be utilized to excellent advantage. However, although many producer-gas plants in daily operation are using bituminous coal and lignite, many power-plant men are still skeptical regarding the successful commercial use of these fuels.

## PRESENT STATUS OF PRODUCER-GAS PLANTS.

In this connection the author can offer no clearer idea of the present status of the producer-gas power plant than is presented in his letter of March 4, 1915, to the prime movers committee of the National Electric Light Association in response to general queries from the committee. The text of the letter follows:

In response to your request for a brief summary of the producer-gas situation, I take pleasure in submitting herewith a brief outline of some of the more important phases of this development and in answering the queries that you have put to me.

1. *General classification of gas producers.*—Producers for power purposes can be broadly classified as (a) up-draft, (b) down-draft, (c) double-zone.

In the up-draft plant the pressure in the gas generator may be greater or less than that of the atmosphere. If it is greater than that of the atmosphere, the plant is known as an up-draft pressure plant. If it is less than that of the atmosphere, the plant is commonly called an up-draft suction plant, although this reduction of pressure in the gas generator of the larger plants is now usually produced by means of an exhauster instead of by the suction stroke of the engine.

2. *Producers for anthracite coal.*—Little difficulty has been experienced in handling good grades of anthracite coal in gas producers. Occasionally some trouble is experienced owing to the character of the ash or to a low ash-fusing temperature. In the main, however, this fuel has been found satisfactory. For most sections of the country the price of anthracite is relatively too high to warrant its use in plants of large capacity. It is, therefore, largely utilized in plants not exceeding 500 horsepower. As far as I know little has been done in this country with gas producers for the utilization of anthracite screenings or material from the culm pile.

Anthracite coal may be utilized to good advantage in plants of either the up-draft or the down-draft type. Inasmuch as it is comparatively free from tar, anthracite is commonly used in the up-draft producer of either the suction or the pressure type. A single installation of 4,000 horsepower of down-draft producers is using anthracite at \$11.30 a ton in preference to bituminous coal for which the plant was designed. Although the company owns bituminous mines, it places a value of \$8 a ton on its books for the bituminous coal. On this basis of \$8 a ton for the bituminous coal and \$11.30 a ton for the anthracite, a year's operation shows financially in favor of the anthracite. Outside of two or three installations, the individual anthracite plants of this country do not exceed a few hundred horsepower.

3. *Bituminous coal.*—Satisfactory gas producers have been designed for the use of both bituminous coals and lignites of good quality. There is comparatively little difficulty in handling on a commercial scale such plants, provided the fuel is low in ash, has a fairly high ash-fusing temperature, and does not give serious trouble from caking and clinkering. Unfortunately these restrictions are too exacting to fit our common practice in the United States with low-priced fuel. The European situation, where they are able to specify rather definitely the characteristics of the coal, is very different.

My answer, therefore, to your query as to whether producers have been successfully designed for the use of bituminous coals and lignites is "yes" for bituminous coals and lignites of high grades.

I do not say "no" for other grades of bituminous coals and lignites, but I realize that low-grade fuel, high in ash and prone to clinker troubles, is not regarded in the majority of cases as worth the time and effort required. Bituminous coals and lignites of good grade may be successfully used in the up-draft producer if adequate equipment is installed for scrubbing the gas and removing the tar, in the down-draft producer of the continuous type, and in the double-zone producer.

The largest single generator in the United States with which I am familiar is one of 250 square feet of fuel-bed area, burning between 3,000 to 4,000<sup>a</sup> pounds of Illinois bituminous coal per hour. I see no reason why single-shell producers of this type should not be built four times this capacity.

4. My estimate of the horsepower capacity of gas producers installed in the United States for power purposes is as follows:

For anthracite coal:

Plants of more than 500 horsepower rating, horsepower.....	40,000
Plants of less than 500 horsepower rating, horsepower.....	95,000
For bituminous coal, horsepower.....	130,000
For lignite, horsepower.....	15,000

My estimate of the annual fuel consumption of these plants is, roughly:

Anthracite coal, short tons.....	240,000
Bituminous coal, short tons.....	400,000
Lignite coal, short tons.....	60,000

<sup>a</sup> These figures were reported at the time of making the installations. The latest reports (June, 1915) give the fuel-bed area as 210 square feet and the average fuel consumption as 2,750 pounds per hour.

5. In response to your query regarding the slow progress in the development of the bituminous-coal producer, I name the following as among the reasons that immediately suggest themselves:

(a) The low price of our fuels. The time has seemingly not yet arrived when there is sufficient demand for reduced operating costs to warrant the investment necessary for the production of producer-gas units of sufficient size to seriously compete with the efficient steam-turbine units of the present day.

(b) As a corollary to *a*, the cost of the labor involved and the reduced capacity limit producers to the utilization of fuels of good grade. As soon as the cost of fuel is sufficiently high to warrant the use of high ash, low-grade material the design of a producer for this purpose will be speedily forthcoming.

(c) Additional difficulties are satisfactory gas cleaning, tar removal or recovery, elimination of sulphur, and the prevention of serious clinkering. All of these difficulties can, I believe, be obviated.

(d) The fact that, up to the present time, no gas engine has been developed of sufficient size to enter into serious competition with the recent steam-turbine units of 30,000 kilowatt and 35,000 kilowatt capacity.

#### PRODUCER-GAS PLANTS IN EUROPE.

In considering the many phases of this problem you will, I believe, be interested to know the trend of European producer-gas practice to-day. I therefore append a few notes gleaned from my inspection last summer in the interest of the United States Bureau of Mines.

We find in Europe a demand for a gas producer to handle all grades of fuel, especially those grades usually sent to the dump. This demand has brought to the European market the revolving, eccentric-grate producer. Among the most important advantages claimed for these producers is automatic ash removal. Dependent on this primary advantage rest the following claims for the revolving grate:

1. Low labor cost for handling ashes.
2. More uniform and more complete combustion.
3. Operation for months without interruption.
4. Ability to handle much more fuel per square foot of fuel-bed area.
5. Less space per 1,000 cubic feet of gas produced or per horsepower of plant.
6. Freedom from dust and the usual excessive heat and dirt during removal of ashes.
7. Production of a gas of nearly uniform quality.
8. Reduction in the cost of upkeep.

If in addition to rotating the grate the grate be placed slightly off center, a feature is introduced that is probably of far greater value in handling high-ash, clinkering fuels than the mere rotation of the grate.

Experience with European fuels has shown that even with the eccentric revolving grate and the usual producer shell construction clinker troubles are not entirely eliminated when poor-grade fuel with low ash-fusing temperature is used. A further important feature—probably the most important single item—for overcoming clinkering and the tendency of the ash to fuse with the producer lining is the water jacketing of the part of the producer shell surrounding the hot zone.

These revolving grate producers are reported to gasify two to three times as much fuel per square foot of fuel-bed area per hour as can be gasified in corresponding up-draft producers with fixed grates.

Claims of very low percentages of carbon in the ash are also made for this type of producer, the reported record for one installation being 5 per cent carbon, or 0.47 per cent of the fuel gasified.



The claims advanced regarding the steam requirements for clinkering coals used in producers with water jackets around the hot zone are to the effect that not over one-quarter as much steam is required as in the jacketless type with fixed grate. The figures given for comparison are 1 pound of steam per pound of fuel for the fixed-grate jacketless producer and 0.29 pound for the revolving eccentric-grate producer. Results with United States coals in fixed-grate jacketless producers indicate that 1 pound of steam per pound of coal is rather high for plants of good size. Seven-tenths of a pound is nearer the figure, although there are undoubtedly many plants, indifferently operated, that are not below the 1-pound rate.

Representatives of companies handling eccentric revolving-grate producers say that they handle coal containing 45 to 55 per cent ash with perfect ease and are satisfied that these producers can meet the conditions required for American high-ash caking coals.

In Europe we also find a great deal of attention given to the by-product gas plant. These plants are not of a few horsepower capacity, but the installations range from 5,000 to 30,000 horsepower. One company alone reports the installation of by-product recovery producer-gas plants using a total of 3,000 tons of fuel per day and aggregating approximately 300,000 horsepower. The capacity and purpose of a few of the larger installations are as follows:

*Capacity and purpose of a few of the larger European producer-gas plants.*

Installation No.	Fuel capacity per day.	Purpose of plant.	Installation No.	Fuel capacity per day.	Purpose of plant.
1	<i>Tons.</i> 320	Recovery of by-products from waste fuels. Gas used for firing boilers and for power.	5	<i>Tons.</i> 135	Power, forge, and plate furnaces, fire-clay kilns, etc.
2	270	Central distributing station.	6	125	Power and firing caustic pots.
3	250	Power and chemical purposes, calcining ore, etc.	7	120	Evaporating brine.
4	150	Special plant for the recovery of by-products. Gas used for firing colliery boilers.	8	120	Power and chemical furnaces.
			9	100	Firing chrome furnaces.
			10	100	Chemical furnaces.

The majority of these plants are used for power development and gas heating; the recovery of by-products, such as sulphate of ammonia, tar, etc., are secondary factors in the operation of the plant. On the other hand, there are several installations in which power is the secondary factor, the plant being run primarily for the valuable by-product, sulphate of ammonia, which brings a commercial return of \$50 to \$60 a ton.

A few plants are operated for the by-products alone. In certain districts in which the manufacturing and industrial interests do not offer a market for the gas the so-called "by-products" become the main products and the true by-product, producer gas, is thrown away. This condition of affairs is peculiarly true in regions in which the fuel runs high in nitrogen. It is reported that an extensive plant of this character is soon to be erected in Africa.

Peat seems to be peculiarly adapted to the requirements for the production of sulphate of ammonia, and several commercial by-product plants using this fuel are now in operation in Europe. Among these are two plants in Italy using, respectively, 140 and 90 tons of peat per day.

One of the most interesting plants visited last summer was a by-product coke plant, in which the coke-oven gas was the main product and the coke the by-product, combined with a by-product producer-gas plant. The coke-oven gas was turned into the city mains for general use. In order to obtain all this gas for sale to the city, a distinct

central producer-gas plant was installed, as the poorer grade gas, which was not available for general city use, was entirely satisfactory for heating the coke ovens. The originator of this combined method says: "The great advantage of this system is the fact that coke breeze and low-grade fuel generally can be used in the producers without lowering the efficiency of the plant." The producer plant consists of five units, each of 20 tons capacity a day. The coke from the ovens, which is regarded as a by-product, finds a ready market for blast-furnace work, and it is estimated that the by-products from the producer plant, sulphate of ammonia and pitch, practically pay the cost of operation of the producer-gas installation. The coal used in the producers is double-screened nut and contains no dust. It is a fine-looking coal and is reported to be high in oils.

The several companies manufacturing by-product gas plants believe that there is a large field for such plants in the United States if the fuels used be carefully selected. They regard our caking coals as bad for this type of plant.

#### SLAGGING GAS PRODUCER.

The blast-furnace type or slagging gas producer has appeared at regular intervals in Europe and investigations along this line have also been conducted in this country both by commercial interests and the United States Bureau of Mines. In spite of the many claims that no such producer can operate with any commercial success, we find at a colliery in Deutsch-Luxemburg a wonderfully interesting installation consisting of four slagging producers. These producers are approximately 10 feet in internal diameter and gasify 60 tons of fuel each per 24-hour day. The first of these producers was installed about two years ago and required much study and many changes. At the present time [March 4, 1915] these producers are working on 10 per cent ash fuel, but the company anticipates utilizing material containing 20 to 30 per cent ash. Several different companies are reported to be using these slagging producers, and the fact that this type of producer has a real commercial position is shown by the catalogue of one of the large producer manufacturers of Germany, which, among several other classifications, lists its producers as—

- (1) Producers with rotating grates.
- (2) Slagging producers.
- (3) Flat-grate producers.
- (4) Step-grate producers.

Considerable interest is also manifested abroad in the use of wood refuse and similar material in gas producers. At the present time, however, these plants are of small size.

The interest of engineers in Europe is also more or less centered on the possibilities of powdered fuel and questions relating to low-temperature distillation. The principal purpose of this latter problem is to procure a clean smokeless fuel for the unlimited number of fireplaces and at the same time to recover from the original fuel large percentages of oils, motor spirits, and ammonia.

In conclusion, I believe that the time is not distant when the price of our coals must necessarily materially increase. When this time does come we will look much more seriously on the important problems connected with our fuel conservation. We will find it imperative to use high-ash low-grade fuels which, in many cases, will have a low ash fusing temperature. Transportation of this high ash material will be out of the question and its conversion into gas at the mines will result. Even though large gas engines are not forthcoming, the economic use of grades of material that can not be utilized directly under steam boilers will warrant large installations for the production of producer gas and its use for steam generation for large turbo-generator units with long-distance electric transmission.

**PRODUCER-GAS PLANT COMPARED WITH STEAM TURBINE.**

The steam turbine naturally lent itself to central-station service. It was a unit easily understood by steam-engine operators in so far as practical operation is concerned; it could be readily erected without radical changes in the boiler-room equipment; and it rapidly met the demand for large central-station units.

The producer-gas plant was an untried factor; it met with strong opposition from the older steam-plant operators, who saw possibilities of being forced out of their positions; its installation meant a complete renovation of the entire plant, with the replacing of the steam boiler by the producer unit. Large units of this type did not materialize, with the natural result that even to-day the producer-gas power plant is not the large central-station unit, although it occupies a strong position among the isolated plants and the small central stations.

Methods of operation vary greatly with local conditions and especially with the personal inclination or opinion of the plant superintendent. This variation is, perhaps, perfectly natural, owing to the newness of producer-gas power, the absence of experienced producer-gas engineers, the absence of cooperation among the operators of these independent isolated plants, and the absence of well-defined commercial investigations.

With this lack of a common interest on the part of the owners or operators of these plants—an interest that has of late tended to place central-station operation on somewhat uniform basis—no standards for comparison of plant operations have appeared and little has been known by the superintendent of one plant regarding the relative efficiency of operation of his own installation and that of another plant of similar type operating under similar conditions.

Several unsuccessful attempts have been made by various organizations to collect data relating to these important matters, but unfortunately little or no information could be had.

Appreciating the value of such data in connection with the many problems of fuel conservation and its investigations into the use of producer-gas power, the Bureau of Mines decided (1) that an effort should be made to procure operating details from a small number of representative producer-gas plants using scrubbed gas either for power or heating purposes; (2) that primarily data from plants using bituminous coals and lignites should be sought, but that data from a few of the larger anthracite plants and the oil and wood plants should be included.

With these points in mind, data sheets were sent to the owners of a limited number of plants, and for the most part the response has been highly gratifying. Returns of a positive character were received from 39 installations.



Although, as might be expected, a great deal of indefiniteness exists regarding many of the operating details, and few, if any, cost data are available, it is believed that in the following pages sufficient detailed information is presented to provoke wholesome discussion on the part of those interested in this form of power; to stimulate responses to future requests from the Bureau of Mines, and to serve as a basis for comparison by the different operators of similar plants.

#### CHARACTER OF FUEL USED IN GAS PRODUCERS.

Owing to the agitation of the Bureau of Mines for several years past concerning the use of high-ash and low-grade fuels, it was hoped that several producer-gas installations would report the use of grades of fuel hitherto regarded as of little commercial value. However, an inspection of several plants in 1913 showed rather conclusively that on the whole the grade of fuel used in producer-gas installations has been steadily rising, and, in the main, the fuel used to-day is decidedly superior to that used six or eight years ago.

Although from the standpoint of the engineers of the Bureau of Mines this condition is to be regretted, it is perhaps perfectly natural, as the tendency of the operators of such plants is to reduce manual labor to a minimum. A feeling of assurance by the management of guaranteed reliability of service is also commercially important in determining the quality of fuel to be used. It is not strange, then, that the highest grade fuel obtainable at a reasonable price is, in nearly every instance, the one adopted.

A rather poor grade of bituminous coal was originally used in a plant of considerable size, but owing to the amount of labor required in operating, the purchasing agent was finally persuaded by the plant superintendent to substitute a high-grade coal at a much higher price per ton.

Such excellent results were at once obtained and the labor cost was so materially reduced that the purchasing agent, who had formerly believed in purchasing the cheapest coal to be had, immediately substituted the high-grade coal throughout the boiler plant as well as the producer plant. The superintendent felt that the greatest service the producer plant had rendered was in improving the grade of fuel used in the steam plant. He stated that he realized that his company was not working toward fuel conservation, but he felt that reliability of service was the uppermost requirement in his plant. He said that he would be glad to use a lower grade fuel at a less cost per ton if he could be shown plants with similar exacting commercial requirements successfully operating on such fuel.

Such comments are, for the most part, typical of the present situation. It is, however, gratifying to note here and there excep-

tions to this tendency. One notable example may be seen in Texas in a plant that uses lignite screenings as the regular fuel for the gas producers. The manager finds these screenings, costing much less than run-of-mine lignite, an excellent producer fuel. He has given a standing order for this material, but has considerable difficulty in procuring enough of it.

As far as is reported, no other operator in that vicinity feels any assurance of success with such screenings as they are generally regarded as inferior fuel.

The high character of the fuels generally used for producer-gas installations is made apparent by the data in the table following.

*Grade of fuel used in different types of gas producers.*

PRODUCERS USING BITUMINOUS COAL.

Plant No.	Ash.	Sulphur.	Heating value per pound as fired.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>B. t. u.</i>
24.....	3.7	0.5	11,540
8.....	6.9	1.6	12,270
12.....	7.4	.5	12,296
17.....	10.1	3.4	12,522
23.....	6.7	2.2	13,741
28.....	.....	.....	13,750
37.....	.....	.....	14,000
33.....	7.0	1.5	14,370
39.....	5.0	.9	14,500
27.....	7.9	1.2	14,535
22.....	4.8	.5	14,769
3.....	7.3	.8	14,800
35.....	5.5	1.0	14,834
25.....	7.0	2.0	.....
20.....	4.7	1.0	.....

PRODUCERS USING LIGNITE.

11.....	7.9	1.0	5,560
4.....	6.9	.6	7,440
11.....	8.2	.4	7,070
31.....	6.0	.....	11,400

PRODUCER USING WOOD.

26.....	.....	.....	5,400
---------	-------	-------	-------

Most lignite contains 30 to 40 per cent water, but, as indicated above, it does not necessarily run high in ash. As regards true fuel conservation, there is seemingly little being done in the United States by the operators of producer-gas plants.

TESTS OF LOW-GRADE FUEL.

It is, perhaps, important again to emphasize the possibilities from high-ash coals. Although it is fully appreciated that commercial conditions make reliability of operation and plant capacity impera-

tive, it is the belief of the author that many of these plants could utilize relatively cheap, poor grades of fuel with an assurance of both reliability and capacity and a net financial gain. The most difficult problem seems to be that of procuring thoroughly competent men for the careful supervision of such installations. The tabulated figures following, showing the results of Bureau of Mines tests, indicate the possibilities of using in gas producers the fuels represented.

*Results of fuel tests, showing adaptability of certain fuels for use in gas producers.*

Source of fuel.	Variety or size of fuel.	Ash.	Moisture.	Quantity of fuel, as fired, consumed in producer per brake horsepower-hour.
		<i>Per cent.</i>	<i>Per cent.</i>	<i>Pounds.</i>
New Mexico.....	Run of mine.....	19.63	3.62	1.10
Tennessee.....	do.....	20.57	3.55	1.45
Iowa.....	.....	20.70	16.69	1.56
Wyoming.....	.....	20.72	9.44	1.70
Do.....	Run of mine.....	21.73	8.65	1.83
Illinois.....	Bone.....	23.12	8.67	2.88
Brazil, South America.....	Run of mine.....	23.44	10.96	2.02
West Virginia.....	Bone.....	28.08	2.91	1.26
Pennsylvania.....	Washery refuse.....	30.35	2.68	2.34
Do.....	do.....	31.89	2.25	2.76
West Virginia.....	Bone.....	43.74	.47	1.65

If these figures are compared with the results on page 25, it will readily be seen that even after due allowance has been made for the difference between the operation of a testing station and that of the average commercial plant, the possibilities from these high-ash fuels warrant thoughtful consideration by the progressive managers of commercial installations.

The manager of a large steel plant in England realizes that his ability to maintain the high-grade qualities of the steel, for which his company has been so long noted, and at the same time to reduce the cost of manufacture to a point below that of his competitors depends on a reduction in the cost of generating power. He is, therefore, seriously attacking the immense refuse heaps that have been accumulating for years at the colliery connected with the steel plant. The material from these refuse heaps is passed through a crusher and then to a washer where approximately one-half of the noncombustible material is removed. The washed fuel, containing approximately 25 per cent of noncombustible material is then utilized in rotary eccentric-grate producers, to the entire satisfaction of the company and with a reduction in the cost of the product of the plant.

In sharp contrast is a plant of several thousand horsepower previously mentioned. This plant is operated in the interests of a large corporation in the United States, which owns extensive bituminous-



coal mines not distant from the plant. The operating company sends its own bituminous coal to the market and uses Pennsylvania anthracite at a cost of over \$11 per ton in its producer-gas plant. Owing to the lower labor charge when anthracite is used, the records of the plant show, for periods of a year or more, that the cost of operation with bituminous coal, if charged on the books at \$8 per ton, is considerably more than with anthracite coal. Although at the prices indicated, the anthracite fuel seems to give the better results, it is a question of no little importance as to whether refuse material from the company's mines might not, even with higher labor costs, prove the least expensive fuel, especially as the plant has ample spare producer capacity.

#### DATA ON FUEL USED AT COMMERCIAL PLANTS.

The Bureau of Mines sent form letters to numerous producer-gas plants, inquiring as to the character of fuel used, the reasons, if any, for having changed fuels, the reasons why certain plants did not use local fuels, and the necessity for a low sulphur content in the coal. The responses may be briefly summarized as follows, the numbers being those assigned to the replies as received. The figures for fixed carbon, volatile matter, moisture, sulphur, and ash are percentages:

##### *Summary of replies to queries regarding operation of producer-gas plants.*

#### UP-DRAFT PLANTS.

##### Plants using anthracite.

1. Fuel, buckwheat. Low-sulphur coal necessary, as high-sulphur coal causes back-firing.

9. Fuel, buckwheat. Pea coal has been used. Buckwheat is fully as cheap and works nearly as well. Low-sulphur coal not necessary for successful operation.

16. Fuel, buckwheat, from  $\frac{3}{8}$ -inch down. Analysis follows:

Fixed carbon.....	82.47
Volatile matter.....	5.29
Moisture.....	.....
Ash.....	12.29

This plant formerly used gas-house coke, but the anthracite works with greater ease, as the coke clinkered badly and gave trouble from tar in the engines.

19. Fuel, Pennsylvania No. 1 buckwheat. Low-sulphur coal regarded as necessary to prevent clinkering.

20. One 200-horsepower producer uses pea anthracite, and one 250-horsepower producer uses run-of-mine bituminous coal. The pea anthracite is used in the spare producer, the other producer having been modified for the use of coke, which is a by-product from the gas department.

36. Fuel, No. 1 buckwheat. Pea anthracite formerly used. Buckwheat costs less and seems to give about the same results.



## Plants using bituminous coal.

## 3. Fuel, Pocohontas slack. Analysis follows:

Fixed carbon.....	73.60
Volatile matter.....	17.70
Moisture .....	1.45
Ash.....	7.25
Sulphur (separately determined).....	.80
British thermal units in coal, as fired.....	14,800

Formerly used Alabama coal. Unsatisfactory. Too much tar. Caused preignition and variable gas quality. Peat is mined in the vicinity. Was tried, mixed with coal, and did fairly well, but gave an offensive, sickening gas. No ill effects from sulphur in coal, except possibly in causing coal-pile fires.

5. Fuel, bituminous coal from Puritan mines, Cambridge, Ohio,  $\frac{3}{4}$ -inch lump. Have experimented with other coals, but the above meets requirements best. Other coal is mined in the vicinity, but it is too high in sulphur. Low-sulphur coal is required, as the sulphur fumes cause distress.

12. Fuel, No. 2 washed nut bituminous coal that passes over a  $\frac{3}{4}$ -inch and through a  $2\frac{1}{2}$ -inch screen. The proximate analysis is:

Fixed carbon.....	54.83
Volatile matter.....	29.91
Moisture.....	7.90
Ash.....	7.36
Sulphur (separately determined).....	.48
British thermal units in coal as fired.....	12,296

Low-sulphur coal is regarded as essential in the operation of this plant to prevent sulphur fumes throughout the factory and a sulphur coating on the manufactured product as the gas is used for annealing.

17. Fuel, Hocking Valley coal in nut sizes passing over 1-inch and through 2-inch screens. The two grades used show—

Fixed carbon.....	51.67	57.60
Volatile matter.....	38.26	34.20
Moisture.....	9.24	4.99
Ash.....	10.07	3.21
Sulphur (separately determined).....	3.42	.73
British thermal units in coal as fired.....	12,522	13,658

The second fuel is reported to give not enough gas and to be too high in by-products.

21. Owing to the fact that the gas is used in connection with furnaces and the products of combustion are discharged into the room, considerable irritation of the lungs and throat is experienced, which seems to cause ill health among the employees unless the percentage of sulphur in the fuel is kept low. The fuel used contains less than 1 per cent sulphur.

25. Fuel, Hocking Valley bituminous nut coal. The analysis follows:

Fixed carbon.....	47.01
Volatile matter.....	36.67
Moisture.....	9.30
Ash.....	6.97
Sulphur (separately determined).....	1.98

The company used anthracite for the first two years, but on the addition of new producers changed to bituminous coal, with a saving due to the difference in price. They find that the present plant operates more easily on bituminous coal than the former producers did on anthracite.

## 33. The analysis of the fuel used follows:

Fixed carbon.....	69.2
Volatile matter.....	21.3
Moisture.....	2.5
Ash.....	7.0
Sulphur (separately determined).....	1.5
British thermal units in coal on dry basis.....	14,760

37. Fuel, bituminous, Westmoreland (Pa.) run-of-mine coal, showing 14,200 British thermal units on the dry basis. Anthracite coal is mined within 50 miles, but the bituminous coal works more freely and is less expensive.

39. Fuel, New River and Pocahontas (W. Va.) coal reduced to extreme fineness before received. Typical analysis of the New River coal follows:

Fixed carbon.....	73.6
Volatile matter.....	20.4
Moisture.....	1.04
Ash.....	4.96
Sulphur (separately determined).....	.91
British thermal units in coal as fired (approximately).....	14,500

Considerable carbon in ash at times because of uneven burning of fuel bed. Have used Pocahontas. No difference noted in general operation between Pocahontas and New River. Some wood is available in the vicinity, but the high cost of procuring it and the uncertainty as to successful gasification in the producers installed prevent its use. No effects, ill or otherwise, have been noted from sulphur.

## Plants using lignite.

## 4. Fuel, Texas, screened, large-lump lignite. Analysis of coal as received:

Fixed carbon.....	27.02
Volatile matter.....	36.92
Moisture.....	29.20
Ash.....	6.86
Sulphur (separately determined).....	.58
British thermal units in coal as received.....	7,442

Obtain better gas with coal of low-sulphur content.

## 11. Fuel, lignite screenings from two mines, showing a proximate analysis as follows:

	Mine 1.	Mine 2.
Fixed carbon.....	17.50	12.13
Volatile matter.....	47.55	48.18
Moisture.....	27.21	31.81
Ash.....	8.21	7.88
Sulphur (separately determined).....	.44	1.03
British thermal units in coal as fired.....	7,073	5,561
British thermal units in coal on dry basis.....	9,717	8,153
British thermal units in coal combustible.....	10,952	9,217

Formerly this plant used lignite lump, but the cheaper grade gives satisfactory results. This cheaper fuel is regarded by the mine operators as of lower grade than the lignite lump. It is lower in heat value and is also of a grade not usually marketable. It works with slightly less ease than the lump lignite, but as it costs less and is really worked with little difficulty it is regarded as a very satisfactory fuel. This lignite is mined in the vicinity of the plant. Low-sulphur fuel is not regarded as essential for successful operation.

38. Fuel, lignite, pea-screenings, near Centralia, Wash. Several other Washington lignites have been tried, but all cost more and were no better or not so good on account of hotter fire and more clinker. The present fuel is of lower grade and highest in ash but works better.

#### Plants using wood.

13. Fuel, waste hardwood in lengths of 1 foot or less, and about 1 to 3 inches in diameter, together with some bark and sawdust. This hardwood runs about 35 to 60 per cent moisture as used. During the early operation of the plant sticks  $1\frac{1}{2}$  to 3 feet long were used, but some difficulty was experienced with arching of the fuel bed and the formation of pockets that burned out the gas.

26. Fuel, cypress refuse from the hog machine. Brought about 1 mile from the mill. Chips and sawdust also used. The wood as fired yielded 5,400 British thermal units, and on dry basis 10,000 British thermal units. Other fuels tried in this plant are: Anthracite, bituminous coal, coke breeze, and pine hog. Coal or coke was found inferior to the wood refuse, as the cypress hog gives a more constant quality of good gas. It works with greater ease and costs less than the other fuels. Low sulphur content is regarded as important because of the deleterious action of  $H_2SO_4$  on the producer plant and engine.

#### DOWN-DRAFT PLANTS.

##### Plants using bituminous coal.

8. Fuel, Big Muddy (Ill.) run-of-mine coal. Analysis follows:

Fixed carbon.....	50.92
Volatile matter.....	32.80
Moisture.....	9.40
Ash.....	6.88
Sulphur (separately determined).....	1.56
British thermal units in dry coal.....	13,553

Satisfactory in price and results. Is regarded as high grade. Makes good gas with little clinker. No trouble has been experienced on account of sulphur.

20. One 200-horsepower producer uses pea anthracite and one 250-horsepower producer uses run-of-mine bituminous coal with—

Fixed carbon.....	76.80
Volatile matter.....	17.11
Moisture.....	1.39
Ash.....	4.70
Sulphur (separately determined).....	1.01

The pea anthracite is used in the spare producer, the other producer having been modified for the use of coke, which is a by-product from the gas department.

22. Uses Pocahontas slack. Analysis follows:

Fixed carbon.....	81.41
Volatile matter.....	13.23
Moisture.....	.54
Ash.....	4.82
Sulphur (separately determined).....	.50
British thermal units in slack as fired.....	14,769

Experiments have been made with other fuels, but no other fuel has been used for any length of time. The lower the sulphur the better the results.



## 23. Fuel, West Virginia bituminous coal run through a 4-inch crusher.

Fixed carbon.....	53.89
Volatile matter.....	36.01
Moisture.....	3.45
Ash.....	6.65
Sulphur (separately determined).....	2.22
British thermal units in coal as fired (average of 49 cars).....	13,741

This company has used other West Virginia coals and Indiana coal but finds the present coal gives a larger producer output. It is of higher grade than the other fuels tried, higher in carbon, lower in ash, and works with greater ease. It costs more than the other fuels.

## 24. Fuel, bituminous southwestern Illinois coal, 1½ to 7 inches. Analysis follows:

Fixed carbon.....	55.51
Volatile matter.....	31.78
Moisture.....	9.02
Ash.....	3.69
Sulphur (separately determined).....	.52
British thermal units in dry coal.....	12,805

Run-of-mine coal from the same mine and a few other similar Illinois coals have been used. The run-of-mine coal was changed because of too much slack, as a more uniform grade of coal is preferred. The present fuel is regarded as of higher grade than that previously used, although it comes from the same mine. It is freer from dirt and of more suitable size for producer use. It operates with greater ease. It costs more per ton at the mine. Low sulphur is not essential but is preferred.

27. Fuel, Pennsylvania run-of-mine bituminous coal. The average analysis of the coal in four cars follows:

Fixed carbon.....	76.94
Volatile matter.....	13.70
Moisture.....	1.47
Ash.....	7.89
Sulphur (separately determined).....	1.18
British thermal units in coal as fired.....	14,535

When this particular grade of coal can not be procured a coal yielding 13,365 British thermal units is substituted. The latter fuel is higher in volatile matter, ash, and sulphur, and lower in fixed carbon. It is not necessary to run the fires quite as hot as with the former fuel on account of the higher volatile matter. The second coal costs less than the first. Low sulphur content is regarded as essential. On account of corrosion the 3-inch steel economizer tubes last only about 18 months.

28. Fuel, West Virginia Pocahontas coal, running from dust to 0.75 inch in size. As fired it yields 13,500 to 14,000 British thermal units, and on the dry basis it yields 14,000 to 14,500 British thermal units.

29. Fuel, Virginia bituminous run-of-mine coal and slack. Washed slack has been used, but a change was made to the fuel first mentioned because no more washed slack could be purchased. The present fuel is regarded as of lower grade, although it works better in the producer and costs less.

## 35. Fuel, Berwyn and White Scalp level, South Fork (Pa.) coal. Analysis:

Fixed carbon.....	77.0
Volatile matter.....	17.0
Moisture.....	.5
Ash.....	5.5
Sulphur (separately determined).....	1.0
British thermal units in coal as fired.....	14,834



This is a higher-grade coal and costs more than that previously used. The change was made as better results can be obtained with the present coal, as it works with greater ease and does not cake or clinker. Low sulphur content is regarded as necessary in order to eliminate the action of sulphuric acid.

#### DOUBLE-ZONE PLANTS.

##### Plants using bituminous coal.

2. Fuel, New Mexico bituminous chestnut coal. Costs less than coal previously used in steam plants. Low-sulphur coal not necessary.

8. Fuel, Big Muddy (Ill.) run-of-mine coal. Analysis follows:

Fixed carbon.....	50.92
Volatile matter.....	32.80
Moisture.....	9.40
Ash.....	6.88
Sulphur (separately determined).....	1.56
British thermal units in coal on dry basis.....	13,553

Satisfactory in price and economy. It is regarded as high grade. Makes good gas with little clinker. No trouble has been experienced on account of sulphur.

##### Plants using lignite.

31. Fuel, lignite, size about 3-inch cubes.

Fixed carbon, approximately.....	40
Volatile matter.....	34
Moisture.....	18
Ash.....	6
British thermal units in coal as fired.....	11,400

#### POUNDS OF FUEL PER SQUARE FOOT OF FUEL-BED AREA PER HOUR.

One of the most important commercial items connected with the design and the operation of gas producers is the determination of the number of pounds of fuel consumed per square foot of fuel-bed area per hour. This rate of fuel consumption varies radically with different types of plants and with different grades and different types of fuel and has led to much difficulty in designing and in rating producers. Early work in this country followed European practice almost entirely and thereby occasioned much trouble in connection with the rating of pioneer plants, and brought about the ultimate failure of many of them. Under certain European conditions fuels of a definite grade are specified, so that high rates of fuel consumption may be obtained. It is not impossible to obtain similar rates of consumption under corresponding circumstances in this country, but as selected fuels are seldom obtainable except for special tests, it has been found that in general in the United States the rate of fuel consumption per square foot of fuel-bed area does not average much over one-half the consumption originally guaranteed by early

manufacturers. This fact has, of course, led to a decided modification in the design of many plants.

Although the possible rate of burning high-grade fuel may be relatively high, much depends on the method of operation and the character of the fuel as to the actual commercial results obtained. Certain types of fuel lend themselves more readily to high rates of fuel consumption than others, and with a given fuel the rate of burning will vary with the nature of the gas-generating process, the down-draft plant consuming practically twice as much fuel per square foot of fuel-bed area per hour as the up-draft. In 1911 the writer<sup>a</sup> stated that down-draft plants were in operation that consumed more than 40 pounds of lignite per square foot of fuel-bed area per hour, and that up-draft producers could gasify comparatively large quantities of fuel per hour for relatively short periods, but that in actual operation with ordinary grades of fuel it was doubtful whether the consumption in the up-draft plants would exceed 15 to 16 pounds per square foot of fuel-bed area per hour. The normal figure at that time was stated to be much nearer 10 pounds, and it was said that 10 pounds would prove to be a high figure for fuels having a large percentage of ash or a sulphur content that tended to produce serious clinkering. It is interesting to compare the figures mentioned with the figures from several plants in commercial operation at the present time, as furnished by the owners or operators of the plants. Such figures are presented in the pages following.

#### FUEL CONSUMPTION OF UP-DRAFT PLANTS USING ANTHRACITE COAL.

The fuel-consumption figures for certain plants using anthracite coal follow:

*Fuel consumption of up-draft gas producers using anthracite coal.*

Plant No.	Horsepower of each gas gen- erator.	Area of fuel bed.	Fuel as fired per square foot of fuel-bed area per hour.	
			As rated.	As actually operated.
		<i>Square feet.</i>	<i>Pounds.</i>	<i>Pounds.</i>
14.....	75	7.1	18.0	.....
9.....	160	12.6	16.0	12.7
34.....	250	38.5	7.8	.....
18.....	300	42.5	8.7	.....
19.....	300	44.2	6.8	7.3
36.....	300	44.2	8.5	4.5
16.....	400	56.6	9.0	.....
Average.....	.....	.....	10.7	.....

<sup>a</sup> Fernald, R. H., and Smith, C. D., *Résumé of producer-gas investigations*: Bull. 13, Bureau of Mines, 1911, p. 45.

The average consumption of the seven plants listed, on the basis of the manufacturers' rating, was 10.7 pounds per square foot of fuel-bed area. The number of plants reporting operating figures for anthracite coal is not enough to permit the presentation of averages, although it is probable that the average figure for commercial operation for up-draft plants using anthracite as a fuel is not far from the average given on the basis of the manufacturers' rating.

#### FUEL CONSUMPTION OF UP-DRAFT PLANTS USING BITUMINOUS COAL.

Figures covering fuel consumption of gas producers using bituminous coal follow:

*Fuel consumption of up-draft gas producers using bituminous coal.*

Plant No.	Horsepower of each gas gen- erator.	Area of fuel bed.	Fuel as fired per square foot of fuel-bed area per hour.	
			As rated.	As actually operated.
		Square feet.	Pounds.	Pounds.
37.....	200	23.7	7.0	7.0
3.....	200	25.0	10.0	7.6
33.....	250	28.3	10.6	7.6
17.....	250	33.1	10.0	13.7
12.....	200	24.0	10.4	7.4
	300	42.0	9.5	
25.....	300	40.0	9.1	8.5
	400	50.0	8.0	
39.....	370	50.2	10.0	6.1
21.....	650	50.0	-----	8.7
	1,000	78.0		
5.....	1,000	70.9	7.0	7.1
12.....	2,500	210.0	14.3	13.1
Average.....			10.4	8.7

For the up-draft plants operating on bituminous coal the range of actual operating rates of fuel consumption seems to be from approximately 6 to more than 13.5 pounds per square foot of fuel-bed area per hour. This marked difference might, on first thought, be supposed to be due to differences in the character of the fuels, but an examination of the records submitted shows that actually the higher-grade fuel is used in the plant reporting the low rate of fuel consumption per square foot of fuel-bed area per hour.

The coal used in the plant reporting the 6-pound rate averaged 14,500 British thermal units as fired and contained 5 per cent ash and 0.9 per cent sulphur, whereas the coal used in the plant reporting the 13.7-pound rate averaged 12,520 British thermal units and contained 10.1 per cent ash and 3.4 per cent sulphur, and the coal in the plant reporting the 13.1-pound rate averaged 12,300 British thermal units and contained 7.4 per cent ash and 0.5 per cent sulphur.

The inference is that the plant with the 6-pound rate was operated much below the capacity of the producer.

Although there seems to be little direct relation between the rate of fuel consumption for the individual plants and the number of pounds of fuel actually burned per square foot of fuel-bed area, the average figures check within reasonable limits and indicate that approximately 8.5 pounds per square foot of fuel-bed area per hour with good-grade bituminous coal is a reasonable figure for an up-draft producer gas plant.

#### FUEL CONSUMPTION OF UP-DRAFT PLANTS USING LIGNITE.

The figures for fuel consumption in gas producers using lignite follow:

*Fuel consumption of up-draft gas producers using lignite.*

Plant No.	Horsepower of each gas gen- erator.	Area of fuel bed.	Fuel as fired per square foot of fuel-bed area per hour.	
			As rated.	As actually operated.
4.....	100	<i>Square feet.</i> 19.6	<i>Pounds.</i> 11.5	<i>Pounds.</i> 11.
7.....	100	20.4	-----	8.
11.....	250	38.5	-----	14.
38.....	300	41.3	10.0	8.
Average.....	-----	-----	10.8	10.

The returns cited for lignite in up-draft plants indicate that approximately 11 pounds of lignite is burned per square foot of fuel-bed area per hour in plants of this type.

Tests at the Government testing station at St. Louis in 1904-1906 indicated that a higher average rate may easily be obtained with lignite in up-draft plants.

#### FUEL CONSUMPTION OF UP-DRAFT PLANTS USING WOOD.

Figures for fuel consumption in gas producers using wood follow:

*Fuel consumption of up-draft gas producers using wood.*

Plant No.	Horsepower of each gas gen- erator.	Area of fuel bed.	Fuel as fired per square foot of fuel-bed area per hour.	
			As rated.	As actually operated.
13.....	150	<i>Square feet.</i> 50.3	<i>Pounds.</i> 12.0	<i>Pounds.</i> -----
26.....	200	50.3	14.0	-----
	280	50.3	-----	14.5



## FUEL CONSUMPTION OF DOWN-DRAFT PLANTS USING BITUMINOUS COAL.

Figures showing fuel consumption in certain plants using bituminous coal follow:

*Fuel consumption of down-draft gas producers using bituminous coal.*

Plant No.	Horsepower of each gas generator.	Area of fuel bed.	Fuel as fired per square foot of fuel-bed area per hour.	
			As rated.	As actually operated.
		<i>Square feet.</i>	<i>Pounds.</i>	<i>Pounds.</i>
22.....	250	17.4	14.4	14.4
27.....	250	17.4	17.0	.....
35.....	250	20.4	14.7	14.7
28.....	375	38.5	13.0	.....
28.....	500	50.0	12.5	.....
8.....	800	36.0	33.4	18.4
23.....	1,500	60.0	27.5	23.5
24.....	1,500	60.0	20.0	16.7
Average.....			19.1	17.5

For the down-draft plants of the double-generator type the figures reported are perhaps not sufficient to warrant conclusive deductions, but when compared with data from other sources they make apparent that the average fuel consumption per square foot of fuel-bed area per hour is nearly correct.

## FUEL CONSUMPTION IN DOWN-DRAFT GAS PRODUCER USING LIGNITE.

Figures procured a few years since from a plant using lignite in 1,100-horsepower units of the down-draft double-generator type indicated a normal fuel rate of 33 pounds per square foot of fuel area per hour for 16 hours a day, and the unusual rate of 48 pounds for 8 hours a day. This instance, however, is too exceptional to be regarded as representative of general practice.

## FUEL CONSUMPTION OF DOUBLE-ZONE PLANTS USING BITUMINOUS COAL.

Figures showing fuel consumption of double-zone gas producers using bituminous coal follow:

*Fuel consumption of double-zone gas producers using bituminous coal.*

Plant No.	Horsepower of each gas generator.	Area of fuel bed.	Fuel as fired per square foot of fuel-bed area per hour.	
			As rated.	As actually operated.
		<i>Square feet.</i>	<i>Pounds.</i>	<i>Pounds.</i>
2.....	250	29.3	24.0	10.3
8.....	500	47.0	17.0	.....

As operating returns were received for only one double-zone plant, no definite direct deductions can be made, but a review of the results of tests of such a plant indicates the rate of fuel consumption under test conditions to be not far from 15 pounds per square foot of fuel-bed area per hour with bituminous coal and 23 pounds with lignite.

#### CONCLUSIONS REGARDING COAL CONSUMPTION.

A study of the operating reports submitted indicates that the average rate of fuel consumption in producer-gas plants is considerably higher than a few years ago. This betterment is due, perhaps, to two reasons—first, a more efficient operation of the plants themselves, and, second, an increasing tendency to use higher grade fuel. The first result is to be commended, but the second should be deplored.

Although too much stress must not be laid on the following values, the deductions from the operating figures supplied above, coupled with personal observation of many other plants, seem to indicate the following approximate values for fuel consumption per square foot of fuel-bed area per hour as representative of good general commercial practice.

*Figures for fuel consumption in gas producers representing good general practice.*

Kind of plant.	Fuel consumption per square foot of fuel-bed area per hour with—							
	Anthracite coal.		Bituminous coal.		Lignite.		Peat.	Wood.
	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.	Average.	Average.
Up-draft using—	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>
Fuel as fired.....	10.0	14.0	8.5	14.0	12.0	17.0	15.0	14.0
Fuel dry.....	10.0	13.5	8.0	13.0	8.5	12.0	12.0	.....
Down-draft using—								
Fuel as fired.....			17.5	23.5	26.5	31.5	35.5	.....
Fuel dry.....			16.5	22.0	18.5	22.0	25.5	.....
Double-zone using—								
Fuel as fired.....			13.5	18.5	21.5	27.0	.....	.....
Fuel dry.....			12.5	17.5	15.0	19.0	.....	.....

#### POUNDS OF FUEL PER HORSEPOWER-HOUR.

Producer-gas investigations of the United States Geological Survey and of the Bureau of Mines, conducted with plants not above the average in efficiency, showed the following approximate fuel consumption per brake horsepower per hour:

*Results of producer-gas investigations, showing fuel consumption per brake horsepower per hour with different fuels.*

Condition of fuel.	Fuel consumption per brake horsepower-hour with—						
	Bituminous coal.			Lignite.			Peat. <sup>a</sup>
	Average.	Maximum.	Minimum.	Average.	Maximum.	Minimum.	Average.
	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>
Fired.....	1.3	2.0	0.8	2.0	2.8	1.5	2.6
Dry.....	1.2	1.8	.8	1.63	2.02	1.35	2.0

<sup>a</sup> One sample only of peat.

Although these figures were procured during the progress of regular tests, the conditions outlined in previous reports of the Bureau of Mines indicate clearly that equally good results should be readily obtained in the average commercial producer-gas plant. With this in mind, the following results, reported by the operators and owners of several plants, are of peculiar interest:

*Results of operation of producer-gas plants showing fuel consumed per brake horsepower-hour with different fuels.*

## ANTHRACITE COAL.

Plant No.	Horse-power of each gas generator.	Total horse-power of plant.	Heat value of fuel as fired.	Fuel fired per brake horsepower-hour.	
				Rating.	Operating.
			<i>B. t. u.</i>	<i>Pounds.</i>	<i>Pounds.</i>
14.....	75	150	.....	1.67	.....
19.....	300	300	.....	1.00	1.30
36.....	300	300	.....	1.25	1.30
16.....	400	400	.....	1.25	.....
18.....	300	600	.....	1.23	.....
9.....	160	640	.....	1.25	1.50
34.....	250	1,500	.....	1.20	1.25
Average.....				1.26	1.34

## BITUMINOUS COAL.

29.....	200	200	.....	1.10	1.00
20.....	250	250	.....	1.25	.....
3.....	200	400	14,800	1.25	2.40
37.....	200	400	14,000	.....	1.65
17.....	250	500	12,520	1.30	.....
33.....	250	500	14,370	1.20	1.00
27.....	250	500	14,535	1.10	.....
22.....	500	500	14,770	1.00	1.20
12.....	{ 200 }	1,000	12,300	1.30	1.40
35.....	{ 300 }	1,000	14,835	1.20	1.00
39.....	{ 370 }	1,100	14,500	1.37	1.80
25.....	{ 300 }	1,400	.....	1.10	1.40
2.....	{ 400 }	1,500	.....	2.80	2.00
28.....	{ 250 }	1,500	.....	.....	.....
8.....	{ 375 }	1,750	13,750	1.29	1.00
23.....	{ 500 }	4,200	12,270	1.60	1.20
24.....	{ 800 }	6,000	13,740	1.10	1.15
24.....	1,500	9,000	11,540	.80	.....
Average.....				1.30	1.40

## LIGNITE.

4.....	100	100	7,440	2.25	2.50
7.....	100	100	.....	.....	2.40
38.....	300	300	.....	.....	2.00
31.....	200	400	11,400	1.50	3.00
Average.....				.....	2.48

## WOOD.

13.....	150	150	.....	4.00	.....
26.....	{ 200 }	1,040	5,400	2.50	3.33
	{ 280 }				

A more direct comparison between the results of commercially operated plants and those obtained at the Government testing station may be had by an inspection of the following tabulation:

*Comparative results of operating producer-gas plants by the Government and by commercial companies.*

Operator.	Fuel consumed per brake horsepower-hour with—										
	Anthracite.			Bituminous coal.			Lignite.			Peat, <sup>a</sup>	Wood <sup>a</sup>
	Average.	Maximum.	Minimum.	Average.	Maximum.	Minimum.	Average.	Maximum.	Minimum.	Average.	Average.
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
Bureau of Mines.....	1.3	1.5	1.3	1.3	2.0	0.8	2.0	2.8	1.5	2.6	.....
Commercial plants.....	1.3	1.5	1.3	1.4	2.4	1.0	2.5	3.0	2.0	.....	3.3

<sup>a</sup> One sample only used.

Figures showing the heat value of the fuel and the actual consumption of fuel as fired per horsepower-hour for different plants follow:

*Heat value of fuel used at different gas producers and fuel consumption per horsepower-hour.*

Plant No.	Heat value of fuel.	Fuel fired per horsepower-hour.	Plant No.	Heat value of fuel.	Fuel fired per horsepower-hour.
	B. t. u.	Pounds.		B. t. u.	Pounds.
4.....	7,440	2.50	37.....	14,000	1.65
31.....	11,400	3.00	33.....	14,370	1.00
8.....	12,270	1.20	22.....	14,770	1.20
12.....	12,300	1.40	3.....	14,800	2.40
23.....	13,740	1.15	35.....	14,835	1.00
28.....	13,750	1.00			

It is at once evident that there is no direct relation between the independent results obtained by different operators and the heat value of the fuel used. This lack of relationship is to be expected, as individual plant conditions are so radically different and producer-gas power is relatively so new that some plants are necessarily much more efficiently operated than others, a fact convincingly borne out by a personal inspection of a large number of plants. Even plants owned by the same corporation, situated in the same district and using the same general grades of fuel, may bear little relation to each other in the results obtained. In one plant after the power used had been changed from steam to producer gas the cost per ton of finished product was reduced to less than one-half of the former cost, whereas in another plant controlled by the same company the cost per ton of finished product was practically doubled after



a similar introduction of producer gas. The difference in the two results seems to have been due to the difference in the methods of the superintendents in charge. One superintendent was a live, wide-awake man who believed it essential to have his power plant kept in perfect condition for the successful manufacture of his product. As a result, he has never shut down for want of power.

The other superintendent felt that the power plant ought to take care of itself. The appearance of the plant testified plainly to his expressed views and serious shutdowns were frequent.

### DEPTH OF FUEL BED.

Data supplied by various producer-gas companies regarding the depth of fuel bed used follow:

*Data regarding depth of fuel bed used in various producer-gas plants.*

#### UP-DRAFT PLANTS.

##### PLANTS BURNING ANTHRACITE COAL.

Plant No.	Grate area.	Approximate quantity of fuel burned per hour.	Depth of bed.
	Square feet.	Pounds.	Feet.
14.....	7.1	125	4
9.....	12.6	160	2.5
6.....	13.6	-----	3
36.....	44.2	200	6
32.....	38.5	-----	2.75
34.....	38.5	300	2
19.....	44.2	300	10
16.....	56.6	500	4

##### PLANTS BURNING BITUMINOUS COAL.

37.....	23.7	165	2
3.....	25.0	190	3
39.....	50.2	300	1.5
25.....	40.0	340	2.5
23.....	50.0	425	2.5
21.....	50.0	435	1.75
17.....	33.1	450	4
5.....	70.9	500	3
21.....	78.0	680	1.75
12.....	210.0	2,750	6.5

##### PLANTS BURNING LIGNITE.

15.....	-----	-----	4
4.....	19.6	225	2
38.....	41.3	350	6.5
11.....	38.5	570	1.5

##### PLANTS BURNING WOOD.

13.....	50.3	605	8
26.....	50.3	725	7

*Data regarding depth of fuel bed used in various producer-gas plants—Continued.*

### DOWN-DRAFT PLANTS.

#### PLANTS BURNING BITUMINOUS COAL.

Plant No.	Grate area.	Approximate quantity of fuel burned per hour.	Depth of bed.
	<i>Square feet.</i>	<i>Pounds.</i>	<i>Feet.</i>
29.....		220	5
22.....	17.4	250	5
27.....	17.4	295	6
35.....	20.4	300	12
28.....	50.0	625	6
8.....	36.0	1,200	8.5
23.....	60.0	1,400	9

### DOUBLE-ZONE PLANTS.

#### PLANT BURNING BITUMINOUS COAL.

2.....	29.3	310	12
--------	------	-----	----

### PLANT BURNING LIGNITE.

31.....			8
---------	--	--	---

### COMPOSITION OF GAS.

The composition of producer gas varies with the type of producer, the methods and skill used in operating it, the uniformity and regulation of the air and steam supply, the kind and quality of fuel used, the depth of fuel bed, the distribution of the fuel, and the uniformity in size of the fuel. The variations in the composition of producer gas under different operating conditions are readily shown by the following results of analyses of the gas produced in different plants.

*Results of analyses of gas from different types of gas producers.*

[Results represent per cent by volume.]

### UP-DRAFT PLANTS.

#### PLANTS BURNING ANTHRACITE COAL.

Plant No.	CO	CH <sub>4</sub>	C <sub>2</sub> H <sub>4</sub>	H <sub>2</sub>	CO <sub>2</sub>	O <sub>2</sub>	N <sub>2</sub>
14.....	24.0	.....	.....	12.0	3.5	0.2	60.3
19.....	21.4	.....	.....	19.0	7.5	.3	51.8
Average.....	22.7	.....	.....	15.5	5.5	.3	56.0

#### PLANTS BURNING BITUMINOUS COAL.

37.....	27.5	2.2	0.7	6.5	2.9	0.2	60.0
17.....	25.0	4.3	.0	14.6	5.2	.9	50.0
39.....	21.8	2.8	.0	11.3	6.4	.4	57.3
25.....	23.6	4.4	.0	14.2	4.9	1.3	51.6
21.....	24.0	4.6	.0	11.6	4.6	.0	55.2
Average.....	24.4	3.7	.1	11.6	4.8	.6	54.8

*Results of analyses of gas from different types of gas producers—Continued.*

**UP-DRAFT PLANTS—Continued.**

**PLANTS BURNING LIGNITE.**

Plant No.	CO	CH <sub>4</sub>	C <sub>2</sub> H <sub>4</sub>	H <sub>2</sub>	CO <sub>2</sub>	O <sub>2</sub>	N <sub>2</sub>
38.....	16.2	5.0	0.0	25.0	6.9	1.3	45.4
11.....	<sup>a</sup> 21.0				6.6	2.5	
11.....	<sup>b</sup> 9.8				16.6	3.2	

**PLANT BURNING WOOD.**

26.....	13.6	8.0	0.0	4.0	12.9	0.0	61.7
---------	------	-----	-----	-----	------	-----	------

**DOWN-DRAFT PLANTS.**

**PLANTS BURNING BITUMINOUS COAL.**

35.....	24.0	1.6	0.0	14.3	2.0	0.2	57.9
8.....	21.2	3.0	.0	9.3	5.0	.3	61.2
24.....	25.6	.0	1.0	13.6	5.0	.0	54.8
Average.....	23.6	1.5	.3	12.4	4.0	.2	58.0

**DOUBLE-ZONE PLANTS.**

**PLANT BURNING BITUMINOUS COAL.**

8.....	23.2	2.8	0.0	11.0	6.2	1.1	55.7
--------	------	-----	-----	------	-----	-----	------

**PLANT BURNING LIGNITE.**

31.....	10.4	3.8	0.0	17.2	15.4	0.4	52.8
---------	------	-----	-----	------	------	-----	------

**OIL-GAS PLANTS.**

**PLANTS BURNING CRUDE OIL.**

Plant No.	CO	CII <sub>4</sub>	H <sub>2</sub> S	C <sub>x</sub> H <sub>y</sub>	H <sub>2</sub>	CO <sub>2</sub>	O <sub>2</sub>	N <sub>2</sub>
10.....	10.2	6.1	0.0	3.8	10.6	6.1	0.0	63.2
30.....	7.4	12.7	3.1	2.6		4.5		69.3

<sup>a</sup> No steam.

<sup>b</sup> With steam.

The averages of several typical analyses of producer gas from the Bureau of Mines testing plant and the averages of the figures presented for plants in commercial operation are given below. Attention is called to the fact that the analyses may not be strictly comparable, because methods of analyses may have differed slightly.

*Average results of analyses of producer gas from Bureau of Mines testing plants and from commercial plants.*

[Results represent per cent by volume.]

UP-DRAFT PLANTS.

Constituent.	Gas from anthracite coal.		Gas from bituminous coal.		Gas from lignite.		Gas from peat. <sup>a</sup>		Gas from wood. <sup>a</sup>	
	Bureau of Mines.	Commercial plants.	Bureau of Mines.	Commercial plants.	Bureau of Mines.	Commercial plants.	Bureau of Mines.	Commercial plants.	Bureau of Mines.	Commercial plants.
Carbon monoxide (CO).....		22.7	18.28	24.4	21.00	.....	21.0	.....		13.6
Methane (CH <sub>4</sub> ).....		.0	3.12	3.7	2.20	.....	2.2	.....		8.0
Ethylene (C <sub>2</sub> H <sub>4</sub> ).....		.0	.18	.1	.40	.....	.4	.....		.0
Hydrogen (H <sub>2</sub> ).....		15.5	12.90	11.6	18.50	.....	18.5	.....		4.0
Carbon dioxide (CO <sub>2</sub> ).....		5.5	9.84	4.8	12.40	.....	12.4	.....		12.9
Oxygen (O <sub>2</sub> ).....		.3	.04	.6	.00	.....	.0	.....		.0
Nitrogen (N <sub>2</sub> ).....		56.0	55.64	54.8	45.50	.....	45.5	.....		61.7

<sup>a</sup> One sample only.

DOWN-DRAFT PLANTS.

Constituent.	Gas from anthracite coal.		Gas from bituminous coal.		Gas from lignite.		Gas from peat.		Gas from wood.	
	Bureau of Mines.	Commercial plants.	Bureau of Mines.	Commercial plants.	Bureau of Mines.	Commercial plants.	Bureau of Mines.	Commercial plants.	Bureau of Mines.	Commercial plants.
Carbon monoxide (CO).....			19.1	23.6	15.0	.....	14.8	.....		
Methane (CH <sub>4</sub> ).....			1.1	1.5	1.7	.....	1.5	.....		
Ethylene (C <sub>2</sub> H <sub>4</sub> ).....			.0	.3	.0	.....	.1	.....		
Hydrogen (H <sub>2</sub> ).....			11.4	12.4	13.3	.....	13.3	.....		
Carbon dioxide (CO <sub>2</sub> ).....			7.6	4.0	11.5	.....	12.9	.....		
Oxygen (O <sub>2</sub> ).....			.2	.2	.2	.....	.6	.....		
Nitrogen (N <sub>2</sub> ).....			60.6	58.0	58.3	.....	56.8	.....		

HEAT VALUE OF GAS.

Figures showing the heat value of the gas from different fuels determined from the average of a large number of tests reported by the Bureau of Mines and also from the figures submitted by the operators of plants in commercial operation are presented below:

*Average heat values of gas from different fuels when used in various types of gas producers.*

[Figures represent British thermal units per cubic foot of gas.]

UP-DRAFT PLANTS.

Location of gas producer.	Gas from anthracite coal.		Gas from bituminous coal.		Gas from lignite.		Gas from peat.		Gas from wood.	
	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.
Bureau of mines testing station.....			152	176	158	188	<sup>a</sup> 175			
Commercial plants.....	138		151	175	157	185			<sup>a</sup> 133	

<sup>a</sup> One sample only.



*Average heat values of gas from different fuels when used in various types of gas producers—Continued.*

## DOWN-DRAFT PLANTS.

Location of gas producer.	Gas from anthracite coal.		Gas from bituminous coal.		Gas from lignite.		Gas from peat.		Gas from wood.	
	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.
Bureau of mines testing station.....			110	123	111	127	<sup>a</sup> 115	<sup>a</sup> 119		
Commercial plants.....			123	130						

## DOUBLE-ZONE PLANTS.

Bureau of mines testing station.....									
Commercial plants.....			( <sup>b</sup> )		118				

## OIL-GAS PLANTS.

Bureau of mines testing station.....									
Commercial plants.....				<sup>c</sup> 215	<sup>c</sup> 230				

<sup>a</sup> Two samples only.

<sup>b</sup> Tests indicate this figure to be approximately 115.

<sup>c</sup> Figures represent two plants burning crude oil.

## CUBIC FEET OF GAS PER POUND OF FUEL.

Tabulated data showing the quantity of producer gas available from various fuels used in the different types of plants follow:

*Average quantities of gas produced from different fuels in different types of gas producers.*

[Figures represent cubic feet of standard (60° F. and 30 inches of mercury) gas per pound of fuel.]

## UP-DRAFT PLANTS.

Location of gas producer.	Gas from bituminous coal.				Gas from lignite.				Gas from peat.			
	As fired.		Dry.		As fired.		Dry.		As fired.		Dry.	
	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.	Average.	Maximum.
Bureau of mines testing station.....	61	101	65	104	36	46	46	53	<sup>a</sup> 30		<sup>a</sup> 38	
Commercial plants.....	75	96										

## DOWN-DRAFT PLANTS.

Bureau of mines testing station.....	65	80	68	82	36	44	52	61	<sup>b</sup> 29	<sup>b</sup> 31	<sup>b</sup> 40	<sup>b</sup> 44
Commercial plants.....	79	82										

<sup>a</sup> One sample only.

<sup>b</sup> Two samples only.

**TIME BETWEEN FUEL CHARGES AND WEIGHT OF FUEL CHARGES.**

Figures showing the time between fuel charges and the weight of fuel charges in the various types of plants follow:

*Time between fuel charges and weight of fuel charges in different types of gas producers.*

**UP-DRAFT PLANTS.****PLANTS BURNING ANTHRACITE COAL.**

Plant No.	Horsepower of each gas generator.	Average fuel burned per hour.	Length of service per day.	Time be- tween fuel charges.	Weight of fuel charge.
		<i>Pounds.</i>	<i>Hours.</i>	<i>Hours.</i>	<i>Pounds.</i>
6.....	150	-----	24	1 to 6	200
9.....	160	160	24	.5	200 to 600
32.....	250	-----	3 to 7	2	400
18.....	300	190	11	24	1,800 to 2,400
19.....	300	300	24	6	1,000 to 1,500
36.....	300	200	10	10	2,000
1.....	350	-----	5 to 24	.5	175

**PLANTS BURNING BITUMINOUS COAL.**

3.....	200	190	24	1	165
37.....	200	165	10	.5	80
17.....	250	450	-----	4	1,500
33.....	250	215	14	.75	250
39.....	370	300	24	.17	100
21.....	{ 650	435	10	.13	100
	{ 1,000	680			
12.....	{ 2,500	2,750	24	2.5	7,000

**PLANTS BURNING LIGNITE.**

4.....	100	225	8	2	500 to 800
7.....	100	175	11	1	175
15.....	100	-----	14	2 to 3	200 to 300
11.....	250	570	24	.33	190
38.....	300	350	24	2 to 3	900

**PLANT BURNING WOOD.**

26.....	{ 200	-----	24	0.5	280
	{ 280				

**DOWN-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

29.....	200	-----	24	0.5	110
27.....	250	295	10	.25	50
35.....	250	300	10	.17	50
28.....	{ 375	625	24	.17	100
	{ 500				
8.....	800	1,200	24	.25	-----
23.....	1,500	1,400	24	.09	-----

**DOUBLE-ZONE PLANTS.****PLANT BURNING BITUMINOUS COAL.**

2.....	250	310	24	1 to 1.5	700
--------	-----	-----	----	----------	-----

## TIME BETWEEN POKINGS OF FUEL BED.

Tabulated data showing the time between pokings of the fuel bed in the various types of plants are presented below:

*Time between pokings of fuel bed in different types of gas producers.*

## UP-DRAFT PLANTS.

## PLANTS BURNING ANTHRACITE COAL.

Plant No.	Area of fuel bed.	Depth of fuel bed.	Approximate amount of fuel burned per hour.	Time between pokings.
	Square feet.	Feet.	Pounds.	Hours.
14.....	7.1	4	125	1
9.....	12.6	2.5	160	.75
6.....	13.6	3	-----	3 to 6
36.....	44.2	6	200	10
34.....	38.5	2	300	1
19.....	44.2	10	300	8
1.....	-----	-----	-----	1
16.....	56.6	4	500	6

## PLANTS BURNING BITUMINOUS COAL.

37.....	23.7	2	165	10
3.....	25.0	3	190	.33
25.....	40.0	2.5	340	4
21.....	50.0	1.75	435	.75
17.....	33.1	4	450	1
33.....	28.3	-----	215	2
5.....	70.9	3	500	1
21.....	78.0	1.75	680	.75
12.....	210	6.5	2,750	12

## PLANTS BURNING LIGNITE.

15.....	-----	4	-----	1
7.....	20.4	-----	175	12
4.....	19.6	2	225	2
11.....	38.5	1.5	570	12

## PLANTS BURNING WOOD.

13.....	50.3	8	605	0.25
26.....	50.3	7	725	.16

## DOWN-DRAFT PLANTS.

## PLANTS BURNING BITUMINOUS COAL.

29.....	-----	5	220	1
22.....	17.4	5	250	(a)
27.....	17.4	6	295	.5 to 1
35.....	20.4	12	300	3
8.....	36.0	8.5	1,200	(a)
23.....	60.0	9	1,400	.5

## DOUBLE-ZONE PLANTS.

## PLANT BURNING BITUMINOUS COAL.

2.....	29.3	12	310	1
--------	------	----	-----	---

## PLANT BURNING LIGNITE.

31.....	-----	8	-----	4
---------	-------	---	-------	---

<sup>a</sup> Not poked.

**TROUBLE FROM HOLES AND CHANNELS IN FUEL.**

Data supplied by owners or operators regarding trouble from holes and channels in the fuel bed in the various types of plants are presented below. The numbers preceding each paragraph refer to plant numbers assigned by the author:

*Data regarding holes and channels in full bed.***UP-DRAFT PLANTS.****Plants burning anthracite coal.**

1. Causes back-firing trouble.
6. Little trouble from holes and channels.
9. Little trouble, because we use sufficient producer capacity.
14. Not serious. Holes easily closed.
16. Considerable stoking required at times.
19. No trouble.
20. Permit gas to burn in producer.
32. Shut the engine down.
34. Make weak gas.
36. No trouble.

**Plants burning bituminous coal.**

3. Cause poor gas and clinker.
5. Little trouble.
12. No trouble when producers have proper attention.
17. No trouble.
21. Cause much trouble.
25. No trouble.
33. Not appreciable.
37. Cause weak gas.
39. Serious. Necessitating laborious work to keep the fire even throughout the bed.

**Plants burning lignite.**

4. Occasional, but never serious.
7. Destroys gas.
11. No trouble. Dropping fuel on fire keeps it down.
38. Very little. Never have this trouble, as little work keeps the fuel bed in fine shape.

**Plants burning wood.**

13. Permit unfixed oxygen to reach top of fuel bed, causing burning.
26. No trouble.

**DOWN-DRAFT PLANTS.****Plants burning bituminous coal.**

8. No trouble.
20. Permits gas to burn in producer.
22. No trouble.
23. Do not let them occur.
24. Readily noticeable on recording calorimeter. Believed to have affected results seriously.
27. Fire must be closely watched and fuel charged lightly.
28. No trouble.
29. Considerable. At times had to shut plant down.
35. No trouble.

**DOUBLE-ZONE PLANTS.****Plants burning bituminous coal.**

2. Quite serious.
8. No trouble.



**METHODS OF OVERCOMING VARIATIONS IN QUALITY OF GAS.**

Data supplied by owners or operators regarding methods of overcoming variations in the quality of the gas yielded by the various types of gas producers follow:

*Data regarding methods used for overcoming variations in gas quality in various types of gas producers.*

**UP-DRAFT PLANTS.****PLANTS BURNING ANTHRACITE COAL.**

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator regarding methods used to overcome variations in quality of gas.
14.....	75	None.
6.....	150	Variation of steam feed.
9.....	160	None.
20.....	200	Suction producer, none; pressure producer, gasometer.
32.....	250	Change setting of mixing valve on engine.
19.....	300	Automatic vapor control and proper cleaning methods.
36.....	300	Automatically controlling steam supply to producer.
16.....	400	Producer automatically regulated.

**PLANTS BURNING BITUMINOUS COAL.**

3.....	200	Gas holder.
37.....	200	By keeping fires regular gas quality varies but very little.
17.....	250	None.
33.....	250	Do.
39.....	370	None, except best possible attention to producer fuel bed.
25.....	300, 400	Hand regulation of air temperature admitted to ash bed.
21.....	650, 1,000	None, except to reprimand gasmen.
5.....	1,000	Keep bed tight, uniform, and free from air pockets.
12.....	200, 300, 2,500	Gas quality checked by recording calorimeter.

**PLANTS BURNING LIGNITE.**

4.....	100	Change mixture at engine.
7.....	100	Adjustment of levers.
15.....	100	More or less free air.
11.....	250	Mixing valves on engines. Gas constant except when cleaning fires.
38.....	300	None. Little variation.

**PLANT BURNING WOOD.**

26.....	200, 280	None.
---------	----------	-------

**DOWN-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

29.....	200	Close attention to fire.
35.....	250	Keep fires and scrubbers clean.
28.....	375, 500	20,000-cubic foot gas holder.
22.....	500	None, except with fires in generators and mixing in gas holders.
8.....	800	Water gas run to regulate temperature of fires.
23.....	1,500	None other than holder.
24.....	1,500	Mixture of gas from different producers. Careful watching of fires.

**DOUBLE-ZONE PLANTS.****PLANT BURNING BITUMINOUS COAL.**

2.....	250	None.
--------	-----	-------

*Data regarding methods used for overcoming variations in gas quality in various types of gas producers—Continued.*

### DOUBLE-ZONE PLANTS—Continued.

#### PLANT BURNING LIGNITE.

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator regarding methods used to overcome variations in quality of gas.
31.....	200	Change mixing valves on engine.

### OIL-GAS PLANT.

#### PLANT BURNING CRUDE OIL.

10.....	400	Variations so slight that governor handles them easily.
---------	-----	---

### METHODS OF CLEANING GAS GENERATORS.

The owners and operators of producer-gas plants were asked to describe briefly their method of cleaning the gas generator; they were requested to state the time and the number of men required for cleaning each unit. Brief abstracts of the replies received are tabulated below.

*Data regarding methods of cleaning gas generators in various types of producer-gas plants.*

### UP-DRAFT PLANTS.

#### PLANTS BURNING ANTHRACITE.

Plant No.	Horsepower of each gas generator.	Total horsepower of plant.	Remarks of owner or operator regarding methods used to clean gas generator.
14.....	75	150	Coal and ash pulled through doors above grate; clinkers removed from walls by bar through poke hole; 3 hours' time of 1 man.
6.....	150	450	Half close valve between producer and gas main; remove all ashes between grates and fire with hoe and poker; poke fire down and add fresh fuel; 1 hour's time of 1 man.
9.....	160	640	Draw ash and clinker, allowing fire to settle down to grate; one-half hour's time of 1 man.
32.....	250	500	After shutting down at 6 p. m., ashes raked out at water seal; 1 hour's time of 1 man.
19.....	300	300	Ash removed from center of fire by rocking grate operated by compressed-air cylinder; fine ash is barred from lining and raked out; 2 hours' time of 1 man.
36.....	300	300	Bar lining; shake grate; 1 hour's time of 1 man.
1.....	350	700	Raked out from bottom; 2 to 3 hours' time of 2 men.
16.....	400	400	Ash is taken out at two intervals in 24 hours; 1 to 2 hours' time of 1 man.

#### PLANTS BURNING BITUMINOUS COAL.

3.....	200	400	Air and steam cut off for a couple of days; ash and coal removed through water seal; 3 hours' time of 1 man.
37.....	200	400	All free ash is removed from bed, leaving fire bridged; bed is then broken down and producer filled; 1½ hours' time of 2 men.
17.....	250	500	Shaking the grates and pulling the ashes; three-fourths of an hour's time of 1 man.
39.....	370	1,100	Remove sufficient ash from water seal to let bed down to normal level; work loose bed down to new level with poke bars; break clinkers off walls and smash fine with chisel bars and heavy hammers; 12 hours' time of 3 men is the minimum.
25.....	300	1,400	(a) Clean by hand-operated rotary table; (b) clean by air-driven shaking grate; 5 minutes' time of 1 man; clean when running.
21.....	650, 1,000	3,650	Not much of this to do.
5.....	1,000	4,000	Draw ashes till bed drops 1 foot; poke fire till solid; 1½ hours' time of 1 man.
12.....	2,500	2,500	Twenty minutes to 1 hour's time of 1 man.

*Data regarding methods of cleaning gas generators in various types of producer-gas plants—*  
Continued.

**UP-DRAFT PLANTS—Continued.****PLANTS BURNING LIGNITE.**

Plant No.	Horse-power of each gas generator.	Total horse-power of plant.	Remarks of owner or operator regarding methods used to clean gas generator.
4.....	100	100	Burn low; rake and clean everything out; 6 hours' time of 2 men.
15.....	100	100	If run 3 days, 24 hours per day, take out everything; if run 14 hours a day, clean every day; for 3 days' run, 3 hours' time of 1 man.
11.....	250	500	Bar fires, and take out ashes, working around producer several times to prevent fire dropping; if gas becomes weak, stop a while; 1½ hours' time of 2 men.
38.....	300	300	Run fuel bed down till 3 or 4 feet deep; open hoppers and all poke holes and draw out fire below; fire is kindled by layer of shavings and fine wood on grates about 12 inches deep; cover with layer of 4 inches of good dry coal; after well lighted, fill with coal; close hoppers and poke holes, starting gas washer for draft; 10 hours' time of 2 or 3 men.

**PLANTS BURNING WOOD.**

13.....	150	150	Producer cleaned daily by removal of ashes; scrubber pit cleaned every hour or so to avoid tar clogging the main; work done by 1 man.
26.....	200, 280	1, 040	Beds lowered by removing ash once a month through water seal; 3 hours' time of 2 men.

**DOWN-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

29.....	200	200	Dig out from bottom and punch down from top; 15 minutes' time of 1 man.
27.....	250	500	Let producer stand from Saturday till Monday; spray fire with water on top for 15 minutes; let stand for 1 hour; clean everything out; recharge with 2,000 pounds of coke; 10 hours' time of 2 men.
35.....	250	1, 000	Fuel and ash are cleaned out every 2 weeks and new fires started; 4 hours' time of 6 men.
28.....	375, 500	1, 750	Clinker sometimes very hard and must be broken with bar from above; 6 to 10 hours' time of 8 men for each pair of producers.
22.....	500	500	All ash, refuse, and clinkers removed; 5 hours' time of 3 men.
8.....	800	3, 200	Set is cut out; cooled with water; ashes removed; clinker cut down; 8 hours' time of 4 men.
23.....	1, 500	6, 000	Water sprayed through charging door directly after run till fire is cooled; coke withdrawn; ashes follow; 4 men, 3 days, 10 hours each.
24.....	1, 500	9, 000	Power-gas producer: Put fire out; remove clinker, coke, and ash; 2 to 4 days' time of 3 to 5 men. Heating-gas producer: Throw fire to one side; clean grate; throw fire back; repeat for other side; 1 to 2 hours' time of 1 man.

**DOUBLE-ZONE PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

2.....	250	250	Not necessary to shut down to clean.
8.....	500	500	See procedure under "Down-draft plants."

**OIL-GAS PLANT.****PLANT BURNING CRUDE OIL.**

30.....	-----	-----	Raise stack valve, shut off blower, remove coke, close door, turn on blast, adjust oil, and lower stack; 10 minutes' time of 1 man.
---------	-------	-------	---

**SERIOUS CLEANING TROUBLES.**

Data furnished by owners and operators regarding cleaning troubles in the different types of gas producers are presented in the tabulation following:

*Data regarding cleaning troubles in various types of gas producers.*

**UP-DRAFT PLANTS.****PLANTS BURNING ANTHRACITE COAL.**

Plant No.	Horsepower of each gas generator.	Total horsepower of plant.	Remarks of owner or operator regarding cleaning troubles.
14.....	75	150	None.
6.....	150	450	None. Only care is to guard against too much intake air.
9.....	160	640	Not to allow ashes and clinker to work up along walls.
32.....	250	500	None.
19.....	300	300	None.
16.....	400	400	Taking out too many ashes, allowing bed to drop too low.

**PLANTS BURNING BITUMINOUS COAL.**

3.....	200	400	Little trouble. If bed hangs it may burn man in falling.
37.....	200	400	Occasional difficulty in breaking down the bridge; then must drive the bar through with the sledge.
39.....	370	1,100	Impossibility of properly breaking up and settling clinker so as to maintain even fire.
5.....	1,000	4,000	None.
12.....	2,500	2,500	None.

**PLANTS BURNING LIGNITE.**

15.....	100	100	Clinker.
11.....	250	500	None unless fire is allowed to drop.
38.....	300	300	None. Same lining in producer for 5 years. Cleaning out hard mixture of tar and ash deposit in gas collector and scrubber inlet pipe is dirty work and takes a little time.

**PLANTS BURNING WOOD.**

13.....	150	150	Filling of gas main by tar. This is burned out occasionally. To be done away with by improved gas cleaner.
26.....	200, 280	1,040	Tar.

**DOWN-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

29.....	200	200	Poor gas and sometimes trouble with clinkers.
27.....	250	500	Producer can not be cleaned while running.
35.....	250	1,000	Heat and dirt.
28.....	375, 500	1,750	Hard clinker.
22.....	500	500	Chopping down clinker.
8.....	800	3,200	None.
23.....	1,500	6,000	None.
24.....	1,500	9,000	Hard clinker. Loss of coke and thin fire to start next run.

**DOUBLE-ZONE PLANT.****PLANT BURNING BITUMINOUS COAL.**

8.....	500	500	None.
--------	-----	-----	-------

**OIL-GAS PLANT.****PLANT BURNING CRUDE OIL.**

30.....	-----	-----	The rapidity with which generator must be cleaned. Reserve gas supply limited.
---------	-------	-------	--



**TIME BETWEEN PERIODS OF DRAWING PRODUCER FIRES.**

The following tabulation gives data regarding time between periods of drawing producer fires in the various types of gas producers:

*Data regarding time between periods of drawing producer fires in various types of gas producers.*

**UP-DRAFT PLANTS.****PLANTS BURNING ANTHRACITE COAL.**

Plant No.	Horsepower of each gas generator.	Total horsepower of plant.	Remarks of owner or operator regarding time between periods of drawing producer fires.
14.....	75	150	20 days.
20.....	200	-----	45 days.
32.....	250	500	Present fire, 8 months; previous fire, 4 years.
19.....	300	300	Once a year.
36.....	300	300	Indefinite.
1.....	350	700	14 days. One producer each week.
16.....	400	400	Not drawn.

**PLANTS BURNING BITUMINOUS COAL.**

3.....	200	400	365 days.
37.....	200	400	10 days.
33.....	250	-----	2 days.
39.....	370	1,100	Fires not drawn except for repairs.
21.....	650, 1,000	3,650	12 months.
5.....	1,000	4,000	Every third day.
12.....	2,500	2,500	6 months to 1 year.

**PLANT BURNING LIGNITE.**

4.....	100	100	120 days.
38.....	300	300	About once a year.

**PLANT BURNING WOOD.**

26.....	200, 280	1,040	3 months or more.
---------	----------	-------	-------------------

**DOWN-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

29.....	200	200	3 months.
20.....	250	-----	45 days.
27.....	250	500	6 days; necessary to cut unit out when coke is 80 to 90 per cent burned out and producer badly clinkered.
35.....	250	1,000	12 days.
28.....	375, 500	1,750	14 days.
22.....	500	500	Weekly.
8.....	800	3,200	10 to 15 days.
23.....	1,500	6,000	13 to 17 days.
24.....	1,500	9,000	15 to 21 days.

**DOUBLE-ZONE PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

2.....	250	250	Do not draw fires.
8.....	500	500	10 to 15 days.

**PLANT BURNING LIGNITE.**

31.....	200	-----	2 years.
---------	-----	-------	----------

**OIL-GAS PLANT.****PLANT BURNING CRUDE OIL.**

30.....	-----	-----	Every 8 hours.
---------	-------	-------	----------------

**METHODS OF SCRUBBING AND CLEANING GAS.**

Data supplied by owners and operators regarding methods of scrubbing and cleaning the gas in the various types of gas producers follow.

*Data regarding methods of scrubbing and cleaning gas in various types of gas producers.*

**UP-DRAFT PLANTS.****PLANTS BURNING ANTHRACITE.**

Plant No.	Horse-power of each gas generator.	Replies from owners or operators.
6.....	150	Counter flow wet coke tower scrubber and excelsior dry scrubber.
20.....	200	Coke.
34.....	250	Water spray dripped through coke.
19.....	300	Static washer.
36.....	300	Static scrubber. Centrifugal tar extractor.
1.....	350	Wood cross sections through which water flows.
16.....	400	Coke between 1 and 2 inches with water sprays.

**PLANTS BURNING BITUMINOUS COAL.**

3.....	200	Latticed spray scrubbers, rotary tar extractors and purifiers.
37.....	200	Rotary water-jet scrubber.
17.....	250	Smith type "F" tar extractors connected in series.
33.....	250	Preliminary wet scrubber, counter current. Rotary scrubber and dry scrubber.
39.....	370	Cooled in first pass of wet scrubber, passed through wooden checker work which is sprayed with water. Passed through centrifugal tar extractor to holder.
25.....	300, 400	Cooling, spraying, and tar extracting. Wood centrifugal tar extractor on one plant and Smith static tar extractor on the other.
21.....	650, 1,000	Saaler washers. Four coolers with water spray. No filling in coolers.
5.....	1,000	Buffalo scrubber. No dry scrubber used. Gas for heating purposes and not for engines.
12.....	<div style="display: inline-block; vertical-align: middle;"> <div style="display: inline-block; vertical-align: middle;"> 200 300 2,500 </div> <div style="display: inline-block; vertical-align: middle; font-size: 3em; line-height: 1;">}</div> </div>	Smith type "F" spun-glass tar extractors.

**PLANTS BURNING LIGNITE.**

4.....	100	Cooling with water and passing through mechanical scrubber.
11.....	250	Gas passes through baffles of 2-inch by 4-inch timbers placed on edge, set close together, 14 feet high. Four streams of water flush each scrubber. Gas then passes through tar extractor.
38.....	300	Spray scrubber and mechanical gas washer (tar extractor).

**PLANTS BURNING WOOD.**

26.....	200, 280	Static scrubber, centrifugal tar extractor.
---------	----------	---

**DOWN-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

29.....	200	Cooled with water spray. Passed through coke with water running over it and down through 6 or 8 feet of straw.
27.....	250	Scrubbers, 28 inches in diameter by 14 feet, each with seven water sprays. Each exhaustor has 16 sprays, throwing water at 40 pounds' pressure against first impeller running at 3,500 revolutions per minute.
35.....	250	Passed through wet and dry scrubber, coke and excelsior.
28.....	375, 500	Passed through wet scrubber and then exhaustor and dry scrubber. One wet scrubber filled with excelsior, other with coke. Both dry scrubbers filled with excelsior.
22.....	500	Cooling in boiler, wet scrubber and dry scrubber.
8.....	800	Static wet scrubbers with water sprays, dry scrubber with excelsior.
23.....	1,500	Gas runs down center of cooler through spray, up inside of shell, through sprays and through a rotary scrubber.
24.....	1,500	Plant A—through vertical boiler, spray cooler, and centrifugal scrubber. Plant B—gas is not clean except from one producer. Special use. Cooled centrifugal tar scrubber and dry cleaner.

*Data regarding methods of scrubbing and cleaning gas in various types of gas producers—Continued.*

**DOUBLE-ZONE PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

Plant No.	Horse-power of each gas generator.	Replies from owners or operators.
2.....	250	Wet scrubber.
8.....	500	Static wet scrubber with water sprays. Dry scrubber with excelsior.

**PLANT BURNING LIGNITE.**

31.....	200	Water spray.
---------	-----	--------------

**OIL-GAS PLANTS.****PLANTS BURNING CRUDE OIL**

10.....	400	Static scrubber filled with baffles cleaning gas to about 1½ grains per cubic foot. Buffalo Forge centrifugal scrubber cleaning to about 0.02 grain per cubic foot. Used 2 years without sign of need of change.
30.....		Gas is forced through three turns, 10 inches by 18 feet, against a gravity flow of water running over baffle plates.

**TIME BETWEEN RENEWALS OF SCRUBBER MATERIAL.**

The owners and operators questioned furnished little information relating to the time between changes in the scrubber material used in the producer-gas installations. The few plants that reported indicated the periods to be as follows:

*Time between renewals of scrubber material in various types of producer-gas plants.*

**UP-DRAFT PLANTS.****Plants burning anthracite coal.**

Plant No.	Time between renewals of scrubber material, months.
20.....	12
16.....	12
6.....	$\left\{ \begin{array}{l} a \text{ 24 to 36} \\ b \text{ 3 to 6} \end{array} \right.$
9.....	6
34.....	48

**Plants burning bituminous coal.**

33.....	8
12.....	2 to 5
21.....	3

**Plant burning lignite.**

11.....	Never.
---------	--------

**Plant burning wood.**

26.....	Never.
---------	--------

*a* Coke.

*b* Excelsior.

## DOWN-DRAFT PLANTS.

## Plants burning bituminous coal.

Plant No.	Time between renewals of scrubber material, months.
29.....	6 to 12
35.....	$\left\{ \begin{array}{l} a \ 12 \\ b \ 0.5 \end{array} \right.$
22.....	$\left\{ \begin{array}{l} c \ 2 \\ d \ 0.5 \end{array} \right.$
8.....	1 $\frac{1}{4}$
24.....	1 to 1 $\frac{1}{2}$
<i>a</i> Coke.	<i>b</i> Excelsior.
<i>c</i> Top.	<i>d</i> Bottom.

## OPINIONS REGARDING SCRUBBING AND CLEANING RESULTS.

The various owners and operators were asked whether the scrubbing and cleaning results in their producers were satisfactory, and if not, why. The data obtained on this point are presented below:

*Data regarding satisfactoriness of scrubbing and cleaning results in various types of gas producers.*

## UP-DRAFT PLANTS.

## PLANTS BURNING ANTHRACITE.

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator regarding satisfactoriness of scrubbing and cleaning results.
6.....	150	Satisfactory.
9.....	160	Do.
20.....	200	Do.
32.....	250	Do.
34.....	250	Do.
19.....	300	Do.
36.....	300	Do.
1.....	350	Fairly.
16.....	400	Satisfactory.

## PLANTS BURNING BITUMINOUS COAL.

3.....	200	Satisfactory.
37.....	200	Very satisfactory.
33.....	250	Not satisfactory. Considerable tar remains in gas and clogs engine valves.
39.....	370	As satisfactory as ever observed in pressure up-draft producer using bituminous coal.
25.....	300, 400	Satisfactory.
21.....	650, 1,000	Does very well, though gas carries tar after cleaning.
5.....	1,000	For heating purposes; not for gas engines.
12.....	$\left\{ \begin{array}{l} 200 \\ 300 \\ 2,500 \end{array} \right.$	$\left. \right\}$ Entirely satisfactory.

## PLANTS BURNING LIGNITE.

4.....	100	Satisfactory.
7.....	100	Do.
11.....	250	Satisfactory. Tar gives no trouble whatever in engine.
38.....	300	Satisfactory. Except occasional cleaning each 3 to 4 months. Tar is of soft paraffin nature, but contains a resinal substance, and when dry is hard to remove.

## PLANTS BURNING WOOD.

13.....	150	Not satisfactory. Are preparing more complete arrangements to recover acids and tars for utilization and for elimination of condensed matter in gas mains.
26.....	200, 280	Not satisfactory. On account of moisture and tar.



*Data regarding satisfactoriness of scrubbing and cleaning results in various types of gas producers—Continued.*

### DOWN-DRAFT PLANTS.

#### PLANTS BURNING BITUMINOUS COAL.

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator regarding satisfactoriness of scrubbing and cleaning results.
29.....	200	Reasonably satisfactory. Slight trouble from tar.
27.....	250	Does not remove lampblack.
35.....	250	Not satisfactory. Considerable dirt and dust get by.
28.....	375, 500	Pipes have to be cleaned out occasionally.
28.....	500	Not entirely satisfactory. Soot and lampblack not completely removed.
8.....	800	Satisfactory.
23.....	1, 500	Entirely satisfactory.
24.....	1, 500	Plant A—satisfactory; plant B—new outfit; good so far (5 months).

### DOUBLE-ZONE PLANTS.

#### PLANTS BURNING BITUMINOUS COAL.

2.....	250	Not satisfactory. Does not remove tar.
8.....	500	Satisfactory.

#### PLANT BURNING LIGNITE.

31.....	200	Satisfactory.
---------	-----	---------------

### OIL-GAS PLANTS.

#### PLANTS BURNING CRUDE OIL.

10.....	400	No tar.
30.....	.....	Not satisfactory. Very difficult to get out the floating carbon.

### MANNER IN WHICH TAR GIVES TROUBLE.

Data obtained from the operators and owners in response to a query as to the way in which tar gave trouble are presented in the tabulation following:

*Data regarding tar troubles in various types of gas producers.*

### UP-DRAFT PLANTS.

#### PLANTS BURNING BITUMINOUS COAL.

Plant No.	Horsepower of each gas generator.	Statement of owner or operator regarding tar troubles.
3.....	200	No operating trouble.
37.....	200	Clogs the governor valve on engine.
33.....	250	Choking and sticking of engine valve.
39.....	370	Clogs mains, fittings, engine regulators, and inlet valve passages. Frequent cleaning of all gas piping required.
25.....	300, 400	No trouble.
21.....	650, 1, 000	Most in gas house in handling. Have little trouble where gas is used.
5.....	1, 000	Chokes pipes between cooling tower and washers.
12.....	{ 200 300 2, 500 }	No trouble at all.

*Data regarding tar troubles in various types of gas producers—Continued.*

**UP-DRAFT PLANTS—Continued.**

**PLANTS BURNING LIGNITE.**

Plant No.	Horse-power of each gas generator	Statement of owner or operator regarding tar troubles.
4.....	100	No serious trouble.
11.....	250	No trouble at all.
38.....	300	No trouble with any part of engine except governor valve chamber and mixing valve chamber. Must be cleaned every 3 months.

**PLANTS BURNING WOOD.**

13.....	150	Clogging mains.
26.....	200, 280	In long gas lines to engine and admission valves.

**DOWN-DRAFT PLANTS.**

**PLANTS BURNING BITUMINOUS COAL.**

29.....	200	Makes engine valve stick and causes some trouble in exhauster.
27.....	250	Produced only when starting new fire. Clogs ports in exhauster.
35.....	250	Clogs scrubbers.
28.....	375, 500	No trouble.
8.....	800	Do.
23.....	1, 500	Do.
24.....	1, 500	Labor for cleaning water seal, stopping sprays in cooler and scrubber. Small amount collects at burners in few months, interfering with the seating of mushroom valves on the line.

**DOUBLE-ZONE PLANTS.**

**PLANTS BURNING BITUMINOUS COAL.**

2.....	250	Sticking intake valves.
8.....	500	No trouble.

**PLANT BURNING LIGNITE.**

31.....	200	On valves of engines.
---------	-----	-----------------------

**OIL-GAS PLANT.**

**PLANT BURNING CRUDE OIL.**

30.....	.....	Causes throttle, poppet, and inlet valves and stack to choke.
---------	-------	---

**TROUBLE CAUSED BY LAMPBLACK.**

Data supplied by owners and operators regarding trouble caused by lampblack follow:

*Data regarding trouble from lampblack in various types of producers.*

**UP-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator regarding trouble caused by lampblack.
3.....	200	Stops water outlet in spray scrubbers.
37.....	200	By collecting in and reducing area of gas mains.
33.....	250	Lampblack mixing with tar.
39.....	370	No trouble observed. If any, it is mixed with tar.
25.....	300, 400	No trouble.
21.....	650, 1,000	Choking flues in producers and coolers.
12.....	200, 300, 2,500	No trouble.

**DOWN-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

27.....	250	Mixes with oil and gums engine valves.
35.....	250	Clogs scrubbers.
28.....	375, 500	Deposits in pipes.
22.....	500	In engine cylinder. Deposits on inlet valves.
8.....	800	No trouble.
24.....	1,500	Do.

**DOUBLE-ZONE PLANT.****PLANT BURNING BITUMINOUS COAL.**

8.....	800	No trouble.
--------	-----	-------------

**OIL-GAS PLANTS.****PLANTS BURNING CRUDE OIL.**

10.....	400	Gradually chokes gas line and soots up gas valves. Requires cleaning every 6 months; 10-hour job.
30.....	.....	Clogs gas main. Works by piston rings and mixes with lubricating oil.

## RELATION OF TAR OR LAMPBLACK IN ENGINE TO COST OF LUBRICATION.

In response to a query as to whether the cost of engine lubrication is materially increased by the presence of tar or lampblack in the engine, the following information was supplied:

*Data regarding relation of lampblack in gas-producer engine to cost of lubrication.*

### UP-DRAFT PLANTS.

#### PLANT BURNING ANTHRACITE COAL.

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator in response to the query: "Is the cost of engine lubrication increased by the presence of tar or lampblack?"
9.....	160	No.

#### PLANTS BURNING BITUMINOUS COAL.

3.....	200	No.
33.....	250	Yes.
39.....	370	Not determined.
25.....	300, 400	No.
21.....	650, 1,000	Not greatly.
5.....	1,000	Yes.

#### PLANT BURNING LIGNITE.

4.....	100	No.
--------	-----	-----

#### PLANT BURNING WOOD.

26.....	200, 280	Yes.
---------	----------	------

### DOWN-DRAFT PLANTS.

#### PLANTS BURNING BITUMINOUS COAL.

29.....	200	No.
22.....	500	Not materially.
8.....	800	No.
23.....	1,500	Yes.
24.....	1,500	Think not.

#### PLANT BURNING LIGNITE.

38.....	300	No. No lampblack ever seen.
---------	-----	-----------------------------

### DOUBLE-ZONE PLANTS.

#### PLANTS BURNING BITUMINOUS COAL.

2.....	250	No.
8.....	500	Do.

#### PLANT BURNING LIGNITE.

31.....	200	Yes.
---------	-----	------

### OIL-GAS PLANTS.

#### PLANTS BURNING CRUDE OIL.

10.....	400	No.
30.....	.....	No. Use cheap oil.



**BURNER TROUBLE CAUSED BY TAR AND LAMPBLACK.**

Data supplied by owners and operators regarding trouble from tar or lampblack clogging the gas burners in the various types of gas producers follow; the number at the beginning of each paragraph indicates the plant that supplied the information :

5. None.
9. None.
12. None.
13. Burner is piece of 8-inch iron pipe; clogged sometimes; cleaned once per month.
17. Latest burner gives no trouble.
22. None.
24. No trouble. Remove some lump tar from burners about once each two months.
25. Gas so cleaned that no trouble is experienced.
30. None.
32. None.
35. Burners clog.
38. None.

**TROUBLE FROM SULPHUR IN PRODUCER GAS.**

Data regarding trouble from sulphur in producer gas as experienced by the various owners and operators follow:

*Data regarding trouble from sulphur in gas of various types of gas producers.*

**UP-DRAFT PLANTS.****PLANTS BURNING ANTHRACITE COAL.**

Plant No.	Horsepower of each gas generator.	Replies from owners or operators.
9.....	160	None.
20.....	200	Do.
18.....	300	Trouble; by clogging passages where velocity of flow is changed and by eating out sheet-steel parts.
19.....	300	None.

**PLANTS BURNING BITUMINOUS COAL.**

3.....	200	Cuts water-cooled valves.
37.....	200	None.
17.....	250	When producer has stood over for 20 hours sulphur is noticed by men at furnaces.
25.....	300, 400	None.
21.....	650, 1,000	Trouble.
5.....	1,000	Trouble; causes distress among employees.
12.....	2,500	Trouble; high sulphur gives unpleasant fumes in factory.

**PLANTS BURNING LIGNITE.**

11.....	250	Exhaust valve seats become pitted.
38.....	300	None, except that engine exhaust has deteriorated roof metal work, guy ropes, etc.

**DOWN-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

29.....	200	None.
20.....	250	Do.
35.....	250	Do.
28.....	375, 500	Do.
22.....	500	Trouble; acts on valves and pipes, and particularly on exhaust.
8.....	800	No serious trouble.
23.....	1,500	None.
24.....	1,500	Do.

**PROPORTION OF SULPHUR ALLOWABLE IN FUEL.**

The owners and operators were asked as to what percentage of sulphur could be present in the gas-producer fuel without causing inconvenience to the workmen. Summarized data supplied in the replies follow:

*Data regarding percentage of sulphur allowable in fuel for producer-gas plants.*

**UP-DRAFT PLANTS.****PLANT BURNING ANTHRACITE COAL.**

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator regarding percentage of sulphur allowable in gas-producer fuel.
19.....	300	1.5 to 1.75 per cent.

**PLANTS BURNING BITUMINOUS COAL.**

17.....	250	3 per cent.
33.....	250	2 per cent.
21.....	650, 1,000	Not over 1 per cent.
12.....	2,500	1.5 per cent.

**PLANT BURNING LIGNITE.**

38.....	300	Do not know per cent, but have seen sulphur plainly visible on fuel and had no trouble.
---------	-----	---

**DOUBLE-ZONE PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

2.....	250	1 per cent has never caused trouble; none higher used.
8.....	500	Approximately 1.5 per cent.

**AUXILIARY POWER USED.**

Many producer plants require a small amount of auxiliary power for the operation of tar extractors, blowers, pumps, exhausters, air compressors, etc. This total auxiliary power varies with the different conditions of plant operation and is, as a rule, less for plants operating with anthracite coal than for plants using tar-producing fuels. An idea of the amount of auxiliary power required may be had from the data presented in the table following, which shows the amount installed and the amount actually used.

*Auxiliary power used at various producer-gas plants.*

## UP-DRAFT PLANTS.

Plant No.	Fuel used.	Total horsepower of plant.	Total horsepower of installed auxiliaries.	Proportion of total plant horsepower—	
				Installed in auxiliaries.	Actually used by auxiliaries.
				<i>Per cent.</i>	<i>Per cent.</i>
38.....	Lignite.....	300	6.0	2.0	1.3
3.....	Bituminous coal.....	400	4.0	1.0	1.0
37.....	do.....	400	18.0	4.5	3.3
16.....	Anthracite coal.....	400	15.0	3.75	-----
11.....	Lignite.....	500	25.0	5.0	5.0
33.....	Bituminous coal.....	500	30.0	6.0	6.0
32.....	Anthracite coal.....	500	10.0	2.0	-----
18.....	do.....	600	15.0	2.5	1.3
12.....	Bituminous coal.....	1,000	17.0	1.7	1.7
26.....	Wood.....	1,040	70.0	6.7	1.9
39.....	Bituminous coal.....	1,100	165.0	15.0	6.2
25.....	do.....	1,400	113.0	8.0	3.9
34.....	Anthracite coal.....	1,500	10.0	0.7	0.7
12.....	Bituminous coal.....	2,500	65.0	2.6	2.6
5.....	do.....	4,000	105.0	2.6	2.0
Average.....	-----	-----	-----	4.3	2.8

## DOWN-DRAFT PLANTS.

29.....	Bituminous coal.....	200	-----	-----	3.0
20.....	do.....	250	5.0	2.0	-----
22.....	do.....	500	10.0	2.0	2.0
27.....	do.....	500	48.0	9.5	5.0
35.....	do.....	1,000	100.0	10.0	10.0
8.....	do.....	3,200	170.0	4.0	2.9
23.....	do.....	6,000	160.0	2.7	2.0
24.....	do.....	9,000	450.0	5.0	1.7
Average.....	-----	-----	-----	5.0	3.8

So much depends on the details of the installation, the number of hours of service per week, and the commercial demands regarding reliability that there may be little relation between the actual auxiliary requirements of different plants. A summary of the data presented in the preceding table follows:

*Summary of data regarding auxiliary power used at various producer-gas plants.*

Kind of plant.	Proportion of total plant power—					
	Installed in auxiliaries.			Actually used by auxiliaries.		
	Maxi-mum.	Mini-mum.	Average.	Maxi-mum.	Mini-mum.	Average.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Up draft.....	15.0	0.7	4.3	6.2	0.7	2.8
Down draft.....	10.0	2.0	5.0	10.0	2.0	3.8

**TROUBLE FROM PRODUCER AUXILIARIES.**

Data regarding serious trouble with producer auxiliaries were requested from the various owners and operators. A summary of the information furnished follows:

*Data regarding trouble from auxiliaries used with gas producers.*

**UP-DRAFT PLANTS.****PLANTS BURNING ANTHRACITE COAL.**

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator regarding serious trouble with producer auxiliaries.
14.....	75	No trouble.
6.....	150	Do.
9.....	160	Do.
20.....	200	Do.
32.....	250	Do.
18.....	300	When fan-type exhausters were used on outlet of producer they were eaten away by sulphurous compounds.
19.....	300	No trouble.
36.....	300	Do.
16.....	400	Do.

**PLANTS BURNING BITUMINOUS COAL.**

3.....	200	No trouble.
37.....	200	Do.
33.....	250	Trouble; scrubber gets choked with tar.
39.....	370	Tar extractors become clogged and passages of scrubbers using excessive power cause high pressures at top of producers, causing gas to leak around poke hole, causing distress to operators. Further, under such conditions flues in fire develop rapidly when pressure is relieved at time poke holes are opened for inspection.
25.....	300, 400	No trouble.
21.....	650, 1,000	Do.
5.....	1,000	Do.
12.....	2,500	No trouble except that due to mechanical defect in steam engine driving the gas pump.

**PLANTS BURNING LIGNITE.**

4.....	100	No trouble.
15.....	100	Do.
11.....	250	Do.
38.....	300	Do.

**PLANTS BURNING WOOD.**

13.....	150	No trouble.
26.....	250	Do.

**DOWN-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

29.....	200	No trouble.
20.....	250	Do.
27.....	250	Trouble on starting new fires a little coal tar is produced, clogging exhauster. Cleaned every 3 weeks.
35.....	250	No trouble.
28.....	375, 500	Do.
22.....	500	Do.
8.....	800	Do.
23.....	1,500	Do.
24.....	1,500	Do.



*Data regarding trouble from auxiliaries used with gas producers—Continued.*

### DOUBLE-ZONE PLANTS.

#### PLANTS BURNING BITUMINOUS COAL.

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator regarding serious trouble with producer auxiliaries.
2.....	250	No trouble.
8.....	500	Do.

#### PLANT BURNING LIGNITE.

31.....	200	No trouble.
---------	-----	-------------

### OIL-GAS PLANTS.

#### PLANTS BURNING CRUDE OIL.

10.....		No trouble.
30.....		No trouble; wash water pump collects tar on runner shaft.

### TIME REQUIRED TO BRING GAS PRODUCERS TO FULL GAS PRODUCTION AFTER STAND-BY.

A summary of the information furnished by owners and operators regarding the time required to bring the various types of producers to condition of normal demand after a stand-by follows:

*Data regarding time required to bring various types of gas producers to condition of normal demand after a stand-by.*

### UP-DRAFT PLANTS.

#### PLANTS BURNING ANTHRACITE COAL.

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator regarding time required for producer to reach full production after stand-by.
6.....	150	1 hour after 18 hours.
32.....	250	$\frac{1}{2}$ hour after 14 hours.
34.....	250	3 hours Monday forenoon.
18.....	300	$\frac{1}{2}$ hour after 13 hours.
19.....	300	1 hour after 14 hours.
36.....	300	40 minutes after 14 hours.
1.....	350	$\frac{1}{2}$ hour.

#### PLANTS BURNING BITUMINOUS COAL.

3.....	200	$\frac{1}{2}$ hour after 12 hours.
37.....	200	1 hour after 14 hours.
17.....	250	1 hour to come in on main line after 10 hours.
33.....	250	1 hour after 12 hours.
39.....	370	1 to 2 hours after 12 hours.
25.....	300, 400	1 hour after 12 hours.
21.....	650, 1,000	$\frac{3}{4}$ hour after 13 hours.
5.....	1,000	1 $\frac{1}{2}$ hours after 24 hours.
12.....	2,500	$\frac{1}{2}$ hour to 1 hour after 14 hours.

*Data regarding time required to bring various types of gas producers to condition of normal demand after a stand-by—Continued.*

**UP-DRAFT PLANTS—Continued.**

**PLANTS BURNING LIGNITE.**

Plant No.	Horsepower of each gas generator.	Remarks of owner or operator regarding time required for producer to reach full production after stand-by.
4.....	100	20 minutes after 12 hours.
7.....	100	1 hour after 12 hours.
11.....	250	2 hours after 12 hours.
38.....	300	20 minutes to $\frac{1}{2}$ hour after 8 hours; 1 hour to $1\frac{1}{2}$ hours after 1 week.

**PLANT BURNING WOOD.**

26.....	200, 280	$\frac{1}{2}$ hour after 24 hours to 1 week.
---------	----------	--

**DOWN-DRAFT PLANTS.**

**PLANTS BURNING BITUMINOUS COAL.**

29.....	200	Service continuous.
27.....	250	Can start engine in 15 minutes after 12 hours, but takes about 1 hour to get to working well.
23.....	1,500	4 hours after 32 hours.
24.....	1,500	30 minutes to 1 hour after 12 hours.

**DOUBLE-ZONE PLANTS.**

**PLANTS BURNING BITUMINOUS COAL.**

2.....	250	$\frac{1}{2}$ hour after 18 hours.
8.....	500	1 hour after 16 hours.

**OIL-GAS PLANT.**

**PLANT BURNING CRUDE OIL.**

30.....	.....	Producer shows good efficiency after 10 minutes' operation.
---------	-------	---

**STAND-BY FUEL.**

Many controversies have arisen regarding the stand-by losses in producer-gas plants. What this percentage may be under test and what it is under commercial operating conditions are two widely different values. Several writers on the subject are in the habit of allowing per stand-by producer hour from 3 to 6 per cent of the fuel charged in the producer per operating hour.

An attempt was made to procure figures covering commercial operation in several plants but the returns were so greatly at variance that no deductions of value could be presented. The figures reported show stand-by percentages ranging from 3 to 33.

## QUANTITY OF WATER USED BY PRODUCER PLANTS.

The quantity of water actually required in the operation of a producer-gas plant and the quantity used by the plants reporting seem to bear little relation to each other. This variance is largely due to the fact that most of the plants reporting seem to have ample water supply in close proximity so that the water cost is small.

Obviously, the quantity of water required in both the generation of the gas and in the cleaning process will vary according to the type of producer plant, the character of the fuel used, the method of operation, and the efficiency of the scrubbing devices.

## QUANTITY OF VAPORIZER WATER USED.

The quantity of vapor used by the gas generators per pound of fuel fired was not reported by the operators queried. Previous investigations have indicated that for up-draft plants this figure usually runs from 0.7 to 1 pound of water for the vaporizer to each pound of anthracite coal fired. With bituminous coals, lignites, peats, wood, and other fuels the quantity of water required by the vaporizer will vary greatly with the percentage of moisture contained in the fuel. With bituminous coals in up-draft plants this figure seems to be not far from that required for anthracite coals. The average for 20 bituminous coals taken at random shows 0.7 pounds of water per pound of coal.

On the other hand, a series of investigations with one grade of fuel in an up-draft plant showed the quality of the gas to vary little for a range of vaporizer consumptions from 0.7 pound to 1.12 pounds.

In using peat it has been found that if the peat contains 25 or 30 per cent moisture no steam is necessary in the operation of the plant.

In one plant in this country operating on lignite the supply of available water is seriously limited. The moisture contained in the lignite is, however, sufficient to more than make up the vaporization requirements so that in the process of cooling the gases enough water vapor is condensed to more than offset the losses. This plant is, therefore, actually increasing its water supply during operation.

During a series of tests with a down-draft producer at the Pittsburgh station of the Bureau of Mines, the average consumption of water by the vaporizer was as follows:

*Consumption of water by vaporizer in down-draft producer of Bureau of Mines.*

Fuel.	Water per pound of fuel as fired, pounds.
Bituminous coal.....	0.23
Lignite.....	.00
Peat.....	.00

## QUANTITY OF SCRUBBER WATER USED.

The quantity of water required for scrubbing the gas in the operation of a producer-gas plant is reported by different writers as follows:

- (a) 1 gallon per 40 cubic feet of gas washed.
- (b) 3.3 gallons per brake horsepower-hour.
- (c) 10 to 15 pounds per pound of fuel used in the gas generator.

If these quantities are reduced to the basis of cubic feet of water required per 1,000 cubic feet of gas washed the range is as follows:

- (a) 3.3 cubic feet per 1,000 cubic feet of gas.
- (b) 6 cubic feet per 1,000 cubic feet of gas.
- (c) 2.6 to 3.9 cubic feet per 1,000 cubic feet of gas.

The latter figures indicate an average of approximately 4 cubic feet of water required for scrubbing 1,000 cubic feet of gas.

During the operation of the testing station of the Bureau of Mines the quantities of scrubber water averaged approximately as follows:

9.7 cubic feet per 1,000 cubic feet of gas with the up-draft plant, including the centrifugal tar extractor.

10.5 cubic feet per 1,000 cubic feet of gas with the down-draft plant.

Figures supplied by the operators of several plants are as follows:

*Quantities of scrubber water used in different gas producers.*

Plant No.	Fuel.	Operation.	Quantity of scrubber water per pound of fuel.
			<i>Pounds.</i>
37.....	Bituminous coal.....	Up-draft.....	76
17.....	do.....	do.....	26
39.....	do.....	do.....	67
25.....	do.....	do.....	36
25.....	do.....	do.....	31
22.....	do.....	Down-draft.....	116
8.....	do.....	Down-draft and double-zone.	95

If these figures are reduced to cubic feet of water per 1,000 cubic feet of gas washed by assuming that 1 pound of bituminous coal is equivalent to 61, 90, and 83 cubic feet of gas for up-draft, down-draft, and double-zone producers, the values are as follows:

*Quantities of scrubber water used per 1,000 cubic feet of gas scrubbed.*

Plant No.	Water per 1,000 cubic feet of gas scrubbed, cubic feet.
37.....	19.9
17.....	6.8
39.....	17.5
25.....	9.5
25.....	8.1
22.....	20.6
8.....	16.9
Average.....	14.2



The range between the temperature of the water entering and that of the water leaving the scrubber seems to depend more on the quantity of water available than on any predetermined temperature as the proper one for the water leaving the scrubber.

The prevailing temperature for the water entering the scrubber is 50° to 90° F., and for the water leaving it 75° to 200° F., the average being about 110° F.

The temperature rise recorded for the different plants is as follows:

*Rise in temperature of scrubber water at different plants.*

Plant No.	Temperature rise, °F.
14.....	45
3.....	8
37.....	10
15.....	50
1.....	155
26.....	37
39.....	44
25.....	45
21.....	70
22.....	15
8.....	55
2.....	10
10.....	40

#### QUANTITY OF WATER REQUIRED TO COOL ENGINE.

The quantity of water required in engine cooling is reported by various writers as follows:

(a) For single-acting engines, 5 to 7 gallons per horsepower-hour.

(b) For large double-acting engines, 4.5 to 5.5 gallons per horsepower-hour for cylinders, stuffing boxes, valves, etc., and 1.75 to 2.25 gallons per horsepower-hour for pistons and pistons rods; or a total ranging from 6.25 to 7.75 gallons per horsepower-hour.

(c) Twenty-five to fifty pounds per brake horsepower-hour, or 3 to 6 gallons.

(d) For a producer-gas engine, single-acting and of less than 200 horsepower, the normal consumption of water per brake horsepower-hour at full load is 5.5 gallons for cooling the engine.

(e) For engines of 2 to 1,000 horsepower, for cylinder covers and stuffing boxes, 5.5 to 6.5 gallons; pistons and rods, 2 to 2.5 gallons; boxes, seats, and exhaust valves, 1 to 1.75 gallons. Total, 9 to 11 gallons per brake horsepower-hour.

(f) With a temperature range of 90° F., 45 pounds or 5.5 gallons will be required per horsepower-hour. In large engines the consumption is smaller, or about 4.25 gallons. To be on the safe side the cooling water may be estimated at 5.5 to 8 gallons.

If the various values be reduced to the common basis of cubic feet of water per horsepower-hour, the figures are as follows:

*Quantity of engine-cooling water required in various plants on basis of cubic feet per horsepower-hour.*

Report.	Cooling water per horsepower-hour, cubic feet.
a.....	0.67 to 0.93
b.....	.83 to 1.03
c.....	.40 to .80
d.....	.73 to .73
e.....	1.20 to 1.47
f.....	.67 to 1.07
Average.....	.75 to 1.00

The Bureau of Mines figures show an average of 0.82 cubic foot per horsepower-hour for a three-cylinder, single-acting engine of 250 horsepower.

The wide variation in practice is shown by the following figures from plant operators:

*Quantity of engine-cooling water required at different producer-gas plants.*

Plant No.	Cooling water per horsepower-hour, cubic feet.
37.....	3.36
25.....	2.80
35.....	2.56
22.....	1.01
8.....	2.18
8.....	2.56

The inlet temperatures reported for the cooling water range from 50° to 90° F. and the outlet temperatures from 86° to 160° F., the average being about 115° F.

#### LUBRICANTS USED.

The quantity of oil required per horsepower-hour varies with the character of the installation and the method of operation. For full load, 24-hour service, the proportion per horsepower-hour is of course greater than for a plant running under light load for a 9-hour or 10-hour day. Some of the figures given by the engine manufacturers for the quantity of engine oil required are as follows:

1. For a 200-horsepower engine, the oil amounts to 1.25 gallons per 10-hour day, or 0.625 gallon per 1,000 horsepower-hours.

2. For a 65-horsepower engine, the oil used is  $\frac{1}{2000}$  gallon per brake horsepower-hour, or 0.500 gallon per 1,000 horsepower-hours.

3. For a 100-horsepower engine, inclosed crank case, 0.5 gallon per 10 hours, or 0.500 gallon per 1,000 horsepower-hours.

4. For a 140-horsepower engine, inclosed crank case, 1 gallon per 10 hours, or 0.715 gallon per 1,000 horsepower-hours.

5. For a 125-horsepower engine, inclosed crank case, 1 quart per 10 hours, or 0.200 gallon per 1,000 horsepower-hours.

The average of these quotations is 0.508 gallon per 1,000 horsepower-hours. The plant operators reported their commercial requirements to be as follows:

*Quantity of oil used in lubricating engines in various gas producers.*

Plant No.	Horsepower of engines.	Length of service per day.	Cylinder oil used per 1,000 horsepower-hours.	Engine oil used per 1,000 horsepower-hours.	Other lubricants used per 1,000 horsepower-hours.
		<i>Hours.</i>	<i>Gallons.</i>	<i>Gallons.</i>	<i>Gallons.</i>
4.....	100	8	2.0		
7.....	100				1.25
14.....			1.8	1.3	
3.....	40, 160	16	2.8		
16.....	190	24		1.0	
11.....	600	24	1.25		
9.....	80, 160, 200, 375	12	1.5	3.0	
12.....	200, 500	24		1.26	
39.....	300	24	.75	.4	.8
25.....	750	10	.5	.17	.07
21.....	125		.13	.4	.1
35.....	150, 250, 300, 600		.5	1.0	
22.....	500	10	.5	.6	.14
23.....	500, 1,000	10	.4	.6	
24.....	300, 2,000	24	2.7	5.3	.7
2.....	115, 300, 750	24	.25	.5	
30.....		24	1.25		
Average.....			1.17	1.11	.51

The average of a number of returns from the operators of reciprocating steam engines indicates the consumption of cylinder oil and engine oil to be approximately the same and to equal 0.13 gallon each per horsepower-hour. On this basis the oil consumption of gas engines seems to be approximately eight or nine times as much as that of reciprocating steam engines. This difference is perhaps not surprising, as the lubricating requirements of the gas engine are much more severe than those of the steam engine, but the ratio seems rather high.

### PREIGNITIONS AND THEIR CAUSES.

Data supplied by owners and operators regarding preignitions and their causes follow. The numbers preceding each paragraph refer to plant numbers assigned by the author.

*Are preignitions frequent and troublesome.—To what do you attribute them?*

#### UP-DRAFT PLANTS.

##### Plants burning anthracite coal.

1. Yes. Carbon deposits.
6. No.
9. No. Most trouble from too high compression.

- 14. No. Excessive hydrogen.
- 16. No.
- 18. No.
- 19. No.
- 20. No. Excessive hydrogen.
- 32. No.
- 34. No. Hydrogen.
- 36. No.

Plants burning bituminous coal.

- 3. No, if good coal is used and valves are tight. Hydrogen, bad gas, and leaky valves.
- 12. No.
- 21. None.
- 25. No. Overhanging fire or hot particles of carbon from lubrication.
- 33. No.
- 37. No.
- 39. No.

Plants burning lignite.

- 4. Yes; at times. Carbon deposits, etc.
- 7. No.
- 11. Only for first hour after starting. Excessive hydrogen.
- 15. No.
- 38. Very seldom. To a hot spot or "chimney" in producer forming high hydrogen or to small particles of tar being sucked off governor valve and holding fire in cylinder. Stopped by throttling gas valve and then sweeping cylinder with air.

Plants burning wood.

- 26. Sticking valves from tar.

DOWN-DRAFT PLANTS.

Plants burning bituminous coal.

- 8. No.
- 20. No. Excessive hydrogen.
- 22. Yes. Dirt, incandescent points, too rich mixture.
- 23. No.
- 24. No.
- 27. Had trouble for a while, but seldom now. Holes or channels in fires.
- 28. After water gas run and when air inlet valve leaks. Too rich gas.
- 29. No. Do not have any.
- 35. No. Carbon.

DOUBLE-ZONE PLANTS.

Plants burning bituminous coal.

- 2. No.
- 8. No.

Plants burning lignite.

- 31. At times. Sticking valves.



## OIL-GAS PLANTS.

Plants burning crude oil.

10. No. Very little. Small portions of lampblack carrying a spark.

30. No. Excess hydrogen under high compression.

## CHANGING OR CLEANING IGNITERS.

The time interval for reliable use of engine igniters is of course more or less dependent on the constituents of the gas, the degree of cleanliness, the amount of sulphur, the engine construction, and the care in operation. The table following indicates the reported practice in this connection:

*Data regarding changing or cleaning of igniters in gas producers.*

## UP-DRAFT PLANTS.

## PLANTS BURNING ANTHRACITE COAL.

Plant No.	Period of operation each day.	Period before igniter is cleaned or changed.
	<i>Hours.</i>	
14.....	24	Cleaned each month; lasts 4 years.
19.....	24	2 months.
16.....	24	$\frac{3}{4}$ month.
6.....	24	$\frac{3}{4}$ month.
9.....	24	$\frac{3}{4}$ month.
1.....	5-24	$1\frac{1}{2}$ months.
34.....	24	$\frac{3}{4}$ month.
18.....	11 $\frac{1}{2}$	Do.

## PLANTS BURNING LIGNITE OR BITUMINOUS COAL.

4.....	8	$\frac{3}{4}$ month.
7.....	11	2 months.
3.....	24	$\frac{3}{4}$ month.
37.....	10	$\frac{3}{4}$ month.
33.....	14	Do.
12.....	24	$\frac{1}{4}$ to $\frac{1}{2}$ month.
39.....	24	2 months.
25.....	10	8 months.
21.....	10	2 months.

## DOWN-DRAFT PLANTS.

## PLANTS BURNING BITUMINOUS COAL.

29.....	24	Years.
27.....	10	$\frac{1}{4}$ month.
35.....	10	$\frac{1}{2}$ month.
22.....	10	4 months.
8.....	24	2 months.
23.....	24	3 months.

## DOUBLE-ZONE PLANTS.

## PLANTS BURNING BITUMINOUS COAL OR LIGNITE.

31.....	24	1 month.
2.....	24	Do.

**GAS PRESSURES AND DISTANCES FOR GAS DELIVERY.**

It is interesting to note that in practically all of these plants the gas is delivered at a short distance from the producer. The reports from 36 plants show a range of distances of 15 to more than 3,000 feet. Twenty-four plants deliver their gas at distances not exceeding 100 feet and only 5 deliver gas to a distance exceeding 500 feet.

None of these plants carries gas at any considerable pressure. The range seems to be from a slightly less than atmospheric to  $2\frac{1}{4}$  pounds higher than atmospheric. In the majority of instances the positive pressure ranges from 1 to 8 ounces.

**RECOVERY OF BY-PRODUCTS FROM WASTE LIQUORS FROM SCRUBBERS, WASHERS, ETC.**

To the query regarding recovery of by-products from waste liquors from scrubbers, washers, etc., the answers were all "None recovered" with the exception of four. Two operators report a special effort to procure the tar; one stated that he recovered the by-products, but gave no further information, and the fourth reported "Not yet."

**COMMERCIAL USE MADE OF TAR PRODUCED.**

The various owners and operators were asked what commercial use was made of the tar produced in their plants and what price was received for it. The data supplied are presented below. The numbers preceding each paragraph refer to plant numbers assigned by the author.

2. None.
3. None.
4. None.
5. None.
7. None.
9. No tar.
11. No market for tar from lignite.
12. Fired under boilers in main boiler plant. Saves 5 tons of coal per day, equal to \$10.
13. Distilled for wood oil, creosote, and pitch.
14. None recovered.
17. Run back into producer.
21. Burned under boiler to make steam for use in gas plant.
23. No tar.
24. Mixed with coal and burned in hand-fired boiler. Not satisfactory.
25. Utilized in steam-boiler plant. The tar from one of the producers returns to the producer and is gasified.
26. Returned to producer by spreading on fuel.
28. None.
30. None.
35. None.
37. Burned under boiler.

38. One barrel per day goes to Puget Sound with scrubber water. Formerly burned with fuel oil under boilers, but caused some trouble on account of water. Will soon sell it or burn it again. Worth about \$1.10 per barrel if burned under boiler.

39. Originally provision was made for introducing the tar into the producer combustion zone, but the tar was too stiff to handle.

#### COMMERCIAL UTILIZATION OF CARBON RESIDUE FROM OIL-GAS PRODUCERS.

Only two replies were received to the query, Is the carbon residue from oil-gas producers commercially utilized, and if so, how? Plant 10 reported that the residue was used under boilers, and plant 30 reported that it was used as fuel in blast furnaces.

#### ENRICHMENT OF PRODUCER GAS.

The various owners and operators were asked whether they knew of any plants that enriched the producer gas. Replies to this inquiry indicated that the enriched producer gas was used to some degree, but not generally. Three or four companies were reported to be using this enriched gas for illuminating purposes. One plant enriched the producer gas with gasoline vapor for use in brazing.

#### TEMPERATURE REQUIRED AT FURNACE.

Data received from the various owners and operators regarding the temperature required at the furnace follow.

10. About 1,400° F.
12. 950 to 1,050° F. in leers.
13. 650° F.
17. 1,700° F.
21. 1,300 to 2,200° F.
24. As high as 1,500° F.
25. 1,500 to 2,000° F.

#### SPARE CAPACITY AND POWER ACTUALLY AVAILABLE.

Practically the same general rules that govern the installation of spare boiler units in steam plants control the installation of spare producer units. The principal points that have to be considered are reliability of plant and continuity of service.

The need of reserve boiler units is, however, much greater under normal operating conditions on account of the necessity of frequent boiler cleaning. Although the intermittent type of producer must be cleaned at regular intervals, depending in length on the percentage of ash in the fuel, the continuous type may be run for years without the fires being drawn if clinker troubles are not excessive.

An examination of the data at hand shows no relation between the daily hours of service and the installation of spare units. Of the plants from which reports were received, ranging in capacity from

100 to 9,000 horsepower, about one-third have reserve units of 20 to 50 per cent of the total installed power.

Ten years ago most gas producers for power purposes were overrated in capacity.

This overrating was the natural result of the blind use of European figures and became for a time a serious matter. Although the rating of such plants is far more conservative to-day, still an examination of the operating reports of 27 plants reveals the fact that approximately one-third of them have considerably less than 100 per cent of their capacity available. One instance is reported in which only 60 per cent of the rated capacity could be realized. The average percentage of available capacity for the plants that are below rating is 75.

On the other hand, 6 of the 27 plants show a capacity considerably above normal rating. The maximum is 155 per cent, and the average for the 6 is 132 per cent.

#### USES OF PRODUCER-GAS PLANTS.

To indicate the varied applications of producer gas and producer-gas power a brief tabulation is presented giving the information received from the 39 companies that reported. The numbers preceding the paragraphs refer to plant numbers.

1. Lights and power (5 to 24 hours per day).
2. Commercial lighting and power (24 hours per day).
3. Manufacture of acid phosphate (24 hours per day).
4. City water company (8 hours per day).
5. General factory purposes; also drying molds and covers.
6. Light and power (24 hours per day).
7. Milling flour (11 hours per day).
8. Induction motor drive (24 hours per day).
9. Lighting and power (24 hours per day).
10. Street and house lighting, ice making, refrigeration, etc. (16 hours per day).
11. Grinding cottonseed cake into meal for export.
12. Motor drives (24 hours per day, 6½ days per week). Gas also used for leers in glass bottling factory.
13. For heating wood-distillation retorts (22 to 24 hours per day).
14. For operating blowing engines and for operating pyrites furnace (24 hours per day).
15. Light and power (14 hours per day).
16. Mining and concentrating iron ore (24 hours per day).
17. Hardening and annealing.
18. Machine shop, electric power (11½ hours per day).
19. Factory lighting and power, motor-driven ammonia compressor for raw-water ice making (summer, 24 hours per day; winter, 8 to 16 hours).
20. Electric lighting and power (24 hours per day).
21. Driving machinery by rope transmission from engine to line shaft (10 hours per day); also forge, annealing, and hardening.
22. Manufacturing (10 hours per day).



23. Manufacture of plate glass, power (24 hours per day, 5½ days per week).
24. Excitation of direct-current electric generator, lights and general plant auxiliary; alternating-current motors driving grinding and polishing machines, 1,000 to 1,200 kilowatts on each alternating-current machine. (Continuous, 24 hours per day, 156 hours per week.) Also melting glass in furnaces and glass annealing.
25. Power in manufacturing plant (10 hours per day). Also hardening and annealing.
26. Power by direct drive and also direct-connected to direct-current generators (7 months, 24 hours per day; 5 months part load, 24 hours per day). Exhaust gas used to produce steam at 60 pounds pressure on vaporizers connected to engine exhausts.
27. Factory power.
28. Electric power for phosphate mining. Principal load is motor-driven centrifugal pumps (156 hours per week).
29. Oil mill, electric power (24 hours per day).
30. Pumping (24 hours per day).
31. Electric power (24 hours per day) and concentration of acid.
32. Machine-shop power (10 hours per day) and drying linings in foundry ladles.
33. Electric shop drive (14 hours).
34. Power in paper mill to drive beaters and pumps; also belted generators (24 hours per day, 6 days per week).
35. Power for manufacturing purposes, shops (10 hours per day); also for heating carbonizing furnaces.
36. Power for manufacturing (10 hours per day).
37. Shop power and lighting (9 hours per day).
38. Driving ammonia compressor and electric generator for power and light (45 months, 24 hours per day with not to exceed one stop per 20 to 30 days for cleaning and adjustments in the summer months; 7 to 8 months, 10 to 20 hours per day). Producer has had fires drawn not to exceed five times in five years.
39. Operating motor-driven centrifugal pumps, rotating and shaking screens and elevators, and for lighting purposes. Also conveyors, rotating dry kilns, washers, shops, etc. (24 hours per day).

### RESPONSE OF PRODUCER TO SUDDEN CHANGES IN DEMAND.

Following are tabulated data relative to the response of various types of gas producers to sudden changes in demand:

*Data regarding response of various types of gas producers to sudden changes in demand.*

#### UP-DRAFT PLANTS.

##### PLANTS BURNING ANTHRACITE COAL.

Plant No.	Horsepower of each gas generator.	Reply of owner or operator to the query, "Does the gas producer respond readily to sudden changes in demand?"
1.....	350	Yes.
6.....	150	Yes; for engine load fluctuation. Starting a second engine takes about 20 minutes before gas is strong.
9.....	160	Reasonably so.
14.....	75	Load is nearly constant. Believe it would be slow.
16.....	400	Yes.
18.....	300	Do.
19.....	300	Do.
20.....	200	Do.
32.....	250	Producer is so large that it instantly covers requirements.
34.....	250	One alone does not. Good on full number.
36.....	300	Yes.

*Data regarding response of various types of gas producers to sudden changes in demand—Continued.*

**UP-DRAFT PLANTS—Continued.**

**PLANTS BURNING BITUMINOUS COAL.**

Plant No.	Horsepower of each gas generator.	Reply of owner or operator to the query, "Does the gas producer respond readily to sudden changes in demand?"
3.....	200	Yes; if fuel bed is in good shape.
5.....	1,000	Yes.
12.....	2,500	Do.
17.....	250	Do.
21.....	650, 1,000	No trouble.
25.....	300, 400	Yes.
33.....	250	No.
37.....	200	Yes; as long as fires are kept regular.
39.....	370	No.

**PLANTS BURNING LIGNITE.**

4.....	100	Yes; when working well.
7.....	100	Yes.
11.....	250	Do.
15.....	100	Do.
38.....	300	Do.

**PLANTS BURNING WOOD.**

13.....	150	Yes; about 5 minutes required for increase in volume.
26.....	200, 280	Yes.

**DOWN-DRAFT PLANTS.**

**PLANTS BURNING BITUMINOUS COAL.**

8.....	800	Perfectly.
20.....	250	Yes.
22.....	500	Have none.
23.....	1,500	Yes.
24.....	1,500	Do.
27.....	250	No.
28.....	375, 500	Proper care required.
29.....	200	No.
35.....	250	Demand is steady.

**DOUBLE-ZONE PLANTS.**

**PLANTS BURNING BITUMINOUS COAL.**

2.....	250	Yes.
8.....	500	Perfectly.

**PLANT BURNING LIGNITE.**

31.....	200	Yes.
---------	-----	------

**PERIOD OF CONTINUOUS OPERATION.**

The data following were furnished in answer to a query regarding the period of operation of gas producers.

*Period of continuous operation of gas producers.***UP-DRAFT PLANTS.****PLANTS BURNING ANTHRACITE COAL.**

Plant No.	Horse-power of each gas generator.	Length of service each day.	Time in continuous operation.
		<i>Hours.</i>	<i>Days.</i>
14.....	75	24	20
6.....	150	24	Continuous.
32.....	250	3 to 7	Do.
34.....	250	24	Do.
19.....	300	24	50
36.....	300	10	Continuous.
1.....	350	5 to 24	7
16.....	400	24	330

**PLANTS BURNING BITUMINOUS COAL.**

17.....	250	-----	6
33.....	250	14	2
25.....	300	10	Continuous.
39.....	400	24	3
21.....	370	24	3
5.....	650	10	Continuous.
12.....	1,000	16	6
	1,000	24	180 to 360
	2,500	24	

**PLANTS BURNING LIGNITE.**

4.....	100	8	200 to 300
7.....	100	11	6
15.....	100	14	3
11.....	250	24	Continuous.
38.....	300	24	Do.

**PLANTS BURNING WOOD.**

13.....	150	-----	7
26.....	200	24	7
	280		

**DOWN-DRAFT PLANTS.****PLANTS BURNING BITUMINOUS COAL.**

29.....	200	24	Continuous.
20.....	250	24	30
27.....	250	10	6
35.....	250	10	6
28.....	250	10	6
	375	24	14
	500		
22.....	500	10	6
8.....	800	24	10 to 15
23.....	1,500	24	15
24.....	1,500	24	15 to 20

**DOUBLE-ZONE PLANTS.****PLANT BURNING BITUMINOUS COAL.**

2.....	250	24	Continuous.
--------	-----	----	-------------

**PLANT BURNING LIGNITE.**

31.....	200	24	730
---------	-----	----	-----

## RELIABILITY AND LENGTH OF SERVICE OF PLANTS.

The following queries were included in the general request for information sent to owners and operators of gas producers: Has the plant proved reliable? If not, what is the cause of failure? How many years has the plant been in service? The data supplied follow.

*Data regarding reliability and length of service of producer-gas plants.*

## UP-DRAFT PLANTS.

## PLANTS BURNING ANTHRACITE COAL.

Plant No.	Horsepower of each gas generator.	Total horsepower of plant.	Reply of owner or operator to queries.
14.....	75	150	Yes. In service 7 years.
19.....	300	300	Yes.
6.....	150	450	In service 3 years.
32.....	250	500	Yes. In service 8 years.
9.....	160	640	Reasonably so. Have had a few minor troubles, but nothing serious. In service 7 years.
1.....	350	700	No. Too many little things that may happen to cause a shutdown, such as change of quality of gas and poor ignition. In service 3½ years.
20.....	200	.....	Yes, after some minor changes.
18.....	300	.....	Yes. In service 4 years.

## PLANTS BURNING BITUMINOUS COAL.

3.....	200	400	Yes, very reliable. Plant competes with a hydroelectric plant that offers current at less than 2 cents per kilowatt-hour. A part of the plant is run with motors, but a majority of it is run from a main line shaft. The manufacturing processes require steady power 24 hours a day, 7 days a week. In service 9 years.
17.....	250	500	Yes.
33.....	250	.....	No. Tar and gas stick valves of engine and choke scrubber and pipes.
39.....	370	1,100	No. Irregularity in thermal quality of gas. Choking of gas mains and engine regulators with tar. Engines are undoubtedly rated too high. Improved mixing valves and gas-inlet passages would probably result in increased capacity. However, considerable expense would be involved. Plant has been superseded by 1,000-kilowatt steam turbine installed in 1911. In service 2 years.
25.....	300, 400	1,400	Yes. One unit in service 3 years, and one unit 2 years.
12.....	200, 300, 2,500	3,500	Yes. Power units in service 3 years; fuel unit, 1 year.
21.....	650, 1,000	3,650	After first year. At first had some trouble on account of tar and with the washing equipment. Since October, 1913, have been using natural gas. In service 4 years.
5.....	1,000	4,000	Producers were operated 3 years continuously and were discontinued, but are still in good condition.

## PLANTS BURNING LIGNITE.

4.....	100	100	Producer was bought for 100 horsepower, but at present does not have capacity of 90.
7.....	100	.....	Yes. In service 5 years.
15.....	100	100	No. Unit has not been run during the past 12 months. Failure thought to be due to prejudice on the part of engineer and helpers, or possibly the attention required was too tedious compared with the steam plant.
38.....	300	300	Yes. Most trouble and expense has been with exhaust piping. This trouble was overcome by installing two short sections of water-jacketed exhaust pipes connecting into fire-brick lined steel shell placed between the twin engines and containing a stack of cast-iron water-heating spiders, or star-shaped sections, which heat all jacket water from 130° to 200° F. About one-half of this hot water is used for industrial purposes and the balance for humidifying the producer ash-pit air. In service 5 years and 4 months.
11.....	250	500	Yes. In service 5 years.



*Data regarding reliability and length of service of producer-gas plants—Continued.*

**UP-DRAFT PLANTS—Continued.**

**PLANTS BURNING WOOD.**

Plant No.	Horsepower of each gas generator.	Total horse-power of plant.	Reply of owner or operator to queries.
13.....	150	150	Yes. After finding out how to use wet wood. When plant was first started used up a large quantity of good dry wood on hand. Plant then put on very green wet wood, requiring much reconstruction of method of operation. In service intermittently for 2 years.
26.....	200, 280	1, 040	Yes.

**DOWN-DRAFT PLANTS.**

**PLANTS BURNING BITUMINOUS COAL.**

29.....	200	200	"All my figures have been destroyed, as we gave up the engine 2 years ago and have since had a fire. We are using hydroelectric power altogether and find it much more successful."
20.....	250	-----	Yes, after some minor changes.
22.....	500	500	Yes. In service 7½ years.
35.....	250	1, 000	Yes. Company is now changing over to hydroelectric power. Producers and engines have been in service over 10 years and for many reasons are not using power economically, owing largely to rapid plant growth and necessarily poor power-plant location.
28.....	375, 500	1, 750	One pair of producers in service 6 years and one pair 2 years.
8.....	800	3, 200	Yes. In service 6 years.
23.....	1, 500	6, 000	Yes. In service 4 years.
24.....	1, 500	9, 000	To certain extent. Lack of knowledge of requirements to make good quality gas, including proper grade and kind of coals, grates, steam and air supply, and method of firing and cost of apparatus.

**DOUBLE-ZONE PLANT.**

**PLANT BURNING BITUMINOUS COAL.**

2.....	250	250	Fairly. Most trouble from poor grade of coal. In service 2½ years.
--------	-----	-----	--



## PUBLICATIONS ON FUEL TECHNOLOGY.

A limited supply of the following publications of the Bureau of Mines is temporarily available for free distribution. Requests for all publications can not be granted, and applicants should limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines, Washington, D. C.

BULLETIN 2. North Dakota lignite as a fuel for power-plant boilers, by D. T. Randall and Henry Kreisinger. 1910. 42 pp., 1 pl., 7 figs.

BULLETIN 4. Features of producer-gas power-plant development in Europe, by R. H. Fernald. 1910. 27 pp., 4 pls., 7 figs.

BULLETIN 5. Washing and coking tests of coal at Denver, Colo., July 1, 1908, to June 30, 1909, by A. W. Belden, G. R. Delamater, J. J. Groves, and K. M. Way. 1910. 62 pp., 1 fig.

BULLETIN 6. Coals available for the manufacture of illuminating gas, by A. H. White and Perry Barker, compiled and revised by H. M. Wilson. 1911. 77 pp., 4 pls., 12 figs.

BULLETIN 16. The uses of peat for fuel and other purposes, by C. A. Davis. 1911. 214 pp., 1 pl., 1 fig.

BULLETIN 24. Binders for coal briquets, by J. E. Mills. 56 pp., 1 fig. Reprint of United States Geological Survey Bulletin 343.

BULLETIN 27. Tests of coal and briquets as fuel for house-heating boilers, by D. T. Randall. 44 pp., 3 pls., 2 figs. Reprint of United States Geological Survey Bulletin 366.

BULLETIN 28. Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo., January 1, 1905, to July 31, 1906; by N. W. Lord. 51 pp. Reprint of United States Geological Survey Bulletin 323.

BULLETIN 31. Incidental problems in gas-producer tests, by R. H. Fernald, C. D. Smith, J. K. Clement, and H. A. Grine. 29 pp., 8 figs. Reprint of United States Geological Survey Bulletin 393.

BULLETIN 33. Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*, by W. T. Ray and Henry Kreisinger. 50 pp., 10 figs. Reprint of United States Geological Survey Bulletin 403.

BULLETIN 36. Alaskan coal problems, by W. L. Fisher. 1911. 32 pp., 1 pl.

BULLETIN 39. The smoke problem at boiler plants, a preliminary report, by D. T. Randall. 31 pp. Reprint of United States Geological Survey Bulletin 334, revised by S. B. Flagg.

BULLETIN 40. The smokeless combustion of coal in boiler furnaces, with a chapter on central heating plants, by D. T. Randall and H. W. Weeks. 188 pp., 40 figs. Reprint of United States Geological Survey Bulletin 373, revised by Henry Kreisinger.

BULLETIN 49. City smoke ordinances and smoke abatement, by S. B. Flagg. 1912. 55 pp.

BULLETIN 55. The commercial trend of the producer-gas power plant in the United States, by R. H. Fernald. 1913. 93 pp., 1 pl., 4 figs.

BULLETIN 56. First series of gold-dust tests in the experimental mine, by G. S. Rice, L. M. Jones, J. K. Clement, and W. L. Egy. 1913. 115 pp., 12 pls., 28 figs.

BULLETIN 58. Fuel-briquetting investigations, July, 1904, to July, 1912, by C. L. Wright. 1913. 277 pp., 21 pls., 3 figs.

BULLETIN 76. United States coals available for export trade, by Van. H. Manning. 1914. 15 pp., 1 pl.

BULLETIN 85. Analyses of mine and car samples of coal collected in the fiscal years 1911 to 1913, by A. C. Fieldner, H. I. Smith, A. H. Fay, and Samuel Sanford. 1914. 444 pp., 2 figs.

BULLETIN 116. Methods of sampling delivered coal and specifications for the purchase of coal for the government, by G. S. Pope. 1916. 64 pp., 5 pls., 2 figs.

TECHNICAL PAPER 2. The escape of gas from coal, by H. C. Porter and F. K. Ovitz. 1911. 14 pp., 1 fig.

TECHNICAL PAPER 5. The constituents of coal soluble in phenol, by J. C. W. Frazer and E. J. Hoffman. 1912. 20 pp., 1 pl.

TECHNICAL PAPER 8. Methods of analyzing coal and coke, by F. M. Stanton and A. C. Fieldner. 1913. 42 pp., 12 figs.

TECHNICAL PAPER 9. The status of the gas producer and of the internal-combustion engine in the utilization of fuels, by R. H. Fernald. 1912. 42 pp., 6 figs.

TECHNICAL PAPER 10. Liquefied products of natural gas; their properties and uses, by I. C. Allen and G. A. Burrell. 1912. 23 pp.

TECHNICAL PAPER 16. Deterioration and spontaneous combustion of coal in storage, a preliminary report, by H. C. Porter and F. K. Ovitz. 1912. 14 pp.

TECHNICAL PAPER 34. Experiments with furnaces for a hand-fired return tubular boiler, by S. B. Flagg, G. C. Cook, and F. E. Woodman. 1914. 32 pp., 1 pl., 4 figs.

TECHNICAL PAPER 35. Weathering of the Pittsburgh coal bed at the experimental mine near Bruceton, Pa., by H. C. Porter and A. C. Fieldner. 1914. 35 pp., 14 figs.

TECHNICAL PAPER 37. Heavy oil as fuel for internal-combustion engines, by I. C. Allen. 1913. 36 pp.

TECHNICAL PAPER 38. Wastes in the production and utilization of natural gas, and methods for their prevention, by Ralph Arnold and F. G. Clapp. 1913. 29 pp.

TECHNICAL PAPER 45. Waste of oil and gas in the Mid-Continent fields, by R. S. Blatchley. 1914. 54 pp., 2 pls., 15 figs.

TECHNICAL PAPER 49. The flash point of oils—methods and apparatus for its determination, by I. C. Allen and A. S. Crossfield. 1913. 31 pp., 2 figs.

TECHNICAL PAPER 50. Metallurgical coke, by A. W. Belden. 1913. 48 pp., 1 pl., 23 figs.

TECHNICAL PAPER 55. The production and use of brown coal in the vicinity of Cologne, Germany, by C. A. Davis. 1913. 15 pp.

TECHNICAL PAPER 57. A preliminary report on the utilization of petroleum and natural gas in Wyoming, by W. R. Calvert, with a discussion of the suitability of natural gas for making gasoline, by G. A. Burrell. 1913. 23 pp.

TECHNICAL PAPER 63. Factors governing the combustion of coal in boiler furnaces; a preliminary report, by J. K. Clement, J. C. W. Frazer, and C. E. Augustine. 1914. 46 pp., 26 figs.

TECHNICAL PAPER 64. The determination of nitrogen in coal, a comparison of various modifications of the Kjeldahl method with the Dumas method, by A. C. Fieldner and C. A. Taylor. 1915. 25 pp., 5 figs.

TECHNICAL PAPER 65. A study of the oxidation of coal, by H. C. Porter. 1914. 30 pp., 12 figs.

TECHNICAL PAPER 74. Physical and chemical properties of the petroleum of California, by I. C. Allen, W. A. Jacobs, A. S. Crossfield, and R. R. Mathews. 1914. 38 pp., 1 fig.

TECHNICAL PAPER 76. Notes on the sampling and analysis of coal, by A. C. Fieldner. 1914. 59 pp., 6 figs.



TECHNICAL PAPER 89. Coal-tar products and the possibility of increasing their manufacture in the United States, by H. C. Porter, with a chapter on coal-tar products used in explosives, by C. G. Storm. 1915. 21 pp.

TECHNICAL PAPER 97. Saving fuel in heating a house, by L. P. Breckenridge and S. B. Flagg. 1915. 35 pp., 3 figs.

TECHNICAL PAPER 104. Analysis of natural gas and illuminating gas by fractional distillation at low temperatures and pressures, by G. A. Burrell, F. M. Seibert, and I. W. Robertson. 1915. 41 pp., 7 figs.

TECHNICAL PAPER 112. The explosibility of acetylene, by G. A. Burrell and G. G. Oberfell. 1915. 15 pp.

TECHNICAL PAPER 123. Notes on the uses of low-grade fuel in Europe, by R. H. Fernald. 1915. 37 pp.

PUBLICATIONS THAT MAY BE OBTAINED ONLY THROUGH THE SUPER-  
INTENDENT OF DOCUMENTS.

The editions for free distribution of the following Bureau of Mines publications are exhausted, but copies may be obtained by purchase from the Superintendent of Documents, Government Printing Office, Washington, D. C., or can be consulted at public libraries. Prepayment of the price is required and should be made in cash (exact amount) or by postal or express money order payable to the Superintendent of Documents.

The Superintendent of Documents is an official of the Government Printing Office and is not connected with the Bureau of Mines.

BULLETIN 1. The volatile matter of coal, by H. C. Porter and F. K. Ovitz. 1910. 56 pp., 1 pl., 9 figs. 10 cents.

BULLETIN 3. The coke industry of the United States as related to the foundry, by Richard Moldenke. 1910. 32 pp. 5 cents.

BULLETIN 7. Essential factors in the formation of producer gas, by J. K. Clement, L. H. Adams, and C. N. Haskins. 1911. 58 pp., 1 pl., 16 figs. 10 cents.

BULLETIN 9. Recent development of the producer-gas power plant in the United States, by R. H. Fernald. 1910. 82 pp., 2 pls. 15 cents. Reprint of United States Geological Survey Bulletin 416.

BULLETIN 11. The purchase of coal by the Government under specifications, with analyses of coal delivered for the fiscal year 1908-9, by G. S. Pope. 80 pp. 10 cents. Reprint of United States Geological Survey Bulletin 428.

BULLETIN 12. Apparatus and methods for the sampling and analysis of furnace gases, by J. C. W. Fraser and E. J. Hoffman. 1911. 22 pp., 6 figs. 5 cents.

BULLETIN 13. Résumé of producer-gas investigations, October 1, 1904, to June 30, 1910, by R. H. Fernald and C. D. Smith. 1911. 393 pp., 12 pls., 250 figs. 65 cents.

BULLETIN 18. The transmission of heat into steam boilers, by Henry Kreisinger and W. T. Ray. 1912. 180 pp., 78 figs. 20 cents.

BULLETIN 14. Briquetting tests of lignite at Pittsburgh, Pa., 1908-9, with a chapter on sulphate-pitch binder, by C. L. Wright. 1911. 64 pp., 11 pls., 4 figs. 15 cents.

BULLETIN 22. Analyses of coals in the United States, with descriptions of mine and field samples collected between July 1, 1904, and June 30, 1910, by N. W. Lord, with chapters by J. A. Holmes, F. M. Stanton, A. C. Fieldner, and Samuel Sanford. 1912. Part I, Analyses, pp. 1-321; Part II, Descriptions of samples, pp. 321-1129. 85 cents.

BULLETIN 23. Steaming tests of coals and related investigations, September 1, 1904, to December 31, 1908, by L. P. Breckenridge, Henry Kreisinger, and W. T. Ray. 1912. 380 pp., 2 pls., 94 figs. 50 cents.

BULLETIN 29. The effect of oxygen in coal, by David White. 80 pp., 3 pls. 20 cents. Reprint of United States Geological Survey Bulletin 382.

BULLETIN 30. Briquetting tests at the United States fuel-testing plant, Norfolk, Va., 1907-8, by C. L. Wright. 41 pp., 9 pls. 15 cents. Reprint of United States Geological Survey Bulletin 385.

BULLETIN 32. Commercial deductions from comparisons of gasoline and alcohol tests on internal-combustion engines, by R. M. Strong. 38 pp. 5 cents. Reprint of United States Geological Survey Bulletin 392.

BULLETIN 34. Tests of run-of-mine and briquetted coal in a locomotive boiler, by W. T. Ray and Henry Kreisinger. 33 pp., 9 figs. 5 cents. Reprint of United States Geological Survey Bulletin 412.

BULLETIN 35. The utilization of fuel in locomotive practice, by W. F. M. Goss. 29 pp., 8 figs. 5 cents. Reprint of United States Geological Survey Bulletin 402.

BULLETIN 41. Government coal purchases under specifications, with analyses for the fiscal year 1909-10, by G. S. Pope, with a chapter on the fuel-inspection laboratory of the Bureau of Mines, by J. D. Davis. 1912. 97 pp., 3 pls., 9 figs. 15 cents.

BULLETIN 43. Comparative fuel values of gasoline and denatured alcohol in internal-combustion engines, by R. M. Strong and Lauson Stone. 1912. 243 pp., 3 pls., 32 figs. 20 cents.

BULLETIN 54. Foundry-cupola gases and temperature, by A. W. Belden. 1913. 29 pp., 3 pls., 16 figs. 10 cents.

BULLETIN 63. Sampling coal deliveries and types of Government specifications for the purchase of coal, by G. S. Pope. 1913. 68 pp., 4 pls., 3 figs. 10 cents.

BULLETIN 97. Sampling and analysis of flue gases, by Henry Kreisinger and F. K. Ovitz. 1915. 68 pp., 1 pl., 37 figs. 15 cents.

TECHNICAL PAPER 1. The sampling of coal in the mine, by J. A. Holmes. 1911. 18 pp., 1 fig. 5 cents.

TECHNICAL PAPER 3. Specifications for the purchase of fuel oil for the Government, with directions for sampling oil and natural gas, by I. C. Allen. 1911. 13 pp. 5 cents.

TECHNICAL PAPER 25. Methods for the determination of water in petroleum and its products, by I. C. Allen and W. A. Jacobs. 1912. 13 pp., 2 figs. 5 cents.

TECHNICAL PAPER 26. Methods for the determination of the sulphur content of fuels, especially petroleum products, by I. C. Allen and I. W. Robertson. 1912. 13 pp., 1 fig. 5 cents.

TECHNICAL PAPER 109. Composition of the natural gas used in 25 cities, with a discussion of the properties of natural gas, by G. A. Burrell and G. G. Oberfell. 1915. 22 pp. 5 cents.

TECHNICAL PAPER 114. Heat transmission through boiler tubes, by Henry Kreisinger and J. F. Barkley. 1915. 36 pp. 23 figs. 10 cents.

TECHNICAL PAPER 115. Inflammability of mixtures of gasoline vapor and air, by G. A. Burrell and H. T. Boyd. 1915. 18 pp., 2 figs. 5 cents.

# INDEX.

## A.

	Page.
Analyses, of Big Muddy, Ill., run-of-mine coal	17, 19
of bituminous coal	15-19
of buckwheat coal	14
of Hocking Valley coal	15
of Illinois bituminous coal	18
of lignite	16, 19
of New River coal	16
of pea anthracite	17
of Pennsylvania coals	18
of Pocahontas slack	15
of producer gas	28-30
of Texas lignite	16
of West Virginia bituminous coal	18
Anthracite coal, advantages of, for gas producers	6
consumption of, per brake horsepower-hour	26
relation of, to fuel-bed area	20-21, 24
operation of gas producers with, cost of	14
pea, analyses of	17
Ash, in gas-producer fuel, percentage of	12, 13
Auxiliaries, for producer-gas plants, horsepower of	49
trouble with	50-51

## B.

Big Muddy, Ill., run-of-mine coal from, analyses of	17, 19
Bituminous coal, analyses of	15-19
consumption of, per brake horsepower-hour	24, 26
relation of, to fuel-bed area	21-24
cost of operation with	14
use of, in gas producers	5, 6
<i>See also</i> Big Muddy, Ill.; Hocking Valley; Illinois; Pennsylvania; Pocahontas slack; West Virginia.	
Bone, tests of	13
Brazil, run-of-mine coal from, tests of	13
Buckwheat coal. <i>See</i> Coal.	
Burners, gas, effect of tar and lampblack on	47
By-products, from gas producers, recovery of	8-9, 60

## C.

Capacity, spare, available at producer-gas plants	61-62
Carbon residue, commercial use of	61
Cleaning of gas producers. <i>See</i> Producer-gas generators.	
Clinkering in gas producers, trouble from	6, 7, 14
Clinkering coals, steam required for	8
Coal, buckwheat, analyses of	14
low-sulphur, necessity for	14, 15, 18
<i>See also</i> Anthracite coal; Bituminous coal; Run-of-mine coal.	
Coal consumption. <i>See</i> Fuel consumption.	
Coke, use of, as by-product	9
Coke breeze, use of, in gas producers	9
Coke-oven gas, production and use of	8-9

## E.

	Page.
Eccentric revolving-grate producers. <i>See</i> Gas producers.	
Engine, cooling of, water for	55-56
lubricating of, cost of	46
oil for	56-57
Europe, gas producers in, demand for	7
producer-gas plants in, fuel capacity of	8
slagging gas producers in	9

## F.

Fires in gas producers, periods of drawing	39
Fuel, conservation of, need for	9, 12
for gas producers, character of	14-19
grades of	9, 11-14
sizes of	13
from refuse heaps, utilization of	13
heating value of, per pound as fired	12
high-ash, use of	9, 11
Fuel bed, area of, fuel consumption relative to	20-23
relation of, to pokings	33
depth of, relation of, to fuel consumption	27-28
relation of, to pokings	33
holes and channels in, trouble from	34
poking of, time between	33
Fuel charges, time between, relation of, to weight	32
Fuel consumption, in gas producers, conclusions regarding	24
rate of	19-27
per brake horsepower-hour	13, 25
per hour in gas producers	32
relation of, to fuel-bed area	20-23
to grate area	27-28
to pokings of fuel bed	33
<i>See also</i> Heating value of fuel.	
Fuel tests, results of	13
Furnace, temperature required at	61

## G.

Gas. <i>See</i> Producer gas.	
Gas generators. <i>See</i> Producer-gas generators.	
Gas-producer products, cost of	13
Gas producers, classification of	5
condition of, after stand-by	51-52
demand for, in Europe	7
down-draft, fuel used in	17-19
eccentric revolving grate, advantages of	7, 8
operation of, period of	64-65
response of, to sudden changes in demand	63-64
time for full production of, after stand-by	51-52
up-draft, fuel used in	14-17
<i>See also</i> Producer-gas generators; Producer-gas plants.	
Germany, gas producers in, types of	9
Grate area, of gas producers, relation of, to fuel consumption	27-28

## H.

Heating value of fuel, per pound as fired	12
relation of, to fuel consumption	26
Heating value of producer gas, from anthracite coal	30, 31
from bituminous coal	30, 31



	Page.		Page.
Heating value of producer gas, from lignite.....	30, 31	Producer gas, from peat, analyses of.....	30
from peat.....	30, 31	heating value of.....	30-31
from wood.....	30, 31	production of.....	31
Hocking Valley coal, analyses of.....	15	from wood, analyses of.....	30
Horsepower, of producer-gas generators.....	20-23,	heating value of.....	30-31
25, 32, 35-48, 50-53		pressures of.....	60
of producer-gas plants.....	25, 36-39, 49, 61-62	scrubbing and cleaning of.....	40-43
I.		variations in, overcoming of.....	35-36
Igniters, engine, changing and cleaning of....	59	Producer-gas generators, cleaning of.....	36-38
<i>See also</i> Preignitions.		horsepower of.....	20-23, 25, 32, 35-48, 50-53
Illinois, bituminous coal from, analyses of....	18	Producer-gas plants, capacity and power	
bone from, tests of.....	13	available at.....	61-62
Iowa, coal from, tests of.....	13	commercial, comparison of, with Govern-	
L.		ment station.....	26
Labor cost, reduction of.....	11	horsepower of.....	25, 36-39, 49, 61-62
Lampblack, effect of, on cost of engine lubri-		number of, in United States.....	5
cation.....	46	reliability and service of.....	66-67
in gas producers, trouble from.....	45, 47	stand-by losses in.....	52
Lignite, analyses of.....	16, 19	water used in.....	53-56
consumption of, per brake horsepower-		<i>See also</i> Gas producers.	
hour.....	24, 26	Producer-gas power, applications of.....	62-63
relation of, to fuel-bed area.....	22-24	R.	
use of, in producer-gas plants.....	5, 6	Run-of-mine coal, tests of.....	13
water in, percentage of.....	12	S.	
Lignite screenings, analyses of.....	16	Scrubber material, renewals of, time between.	41-42
use of, as fuel in gas producers.....	12	Scrubbing of gas. <i>See</i> Producer gas.	
Lubrication of engine, cost of, effect of tar or		Slagging gas producer, use of, in Europe.....	9
lampblack on.....	46	Stand-by. <i>See</i> Gas producers.	
Lubricants, for engine, quantity of.....	56-57	Steam turbine, installation of, in producer-gas	
M.		plants.....	10-11
Moisture, in gas-producer fuel, percentage of.	13	Steel, manufacture of, cost of.....	13
<i>See also</i> Water.		Sulphate of ammonia, from gas producers, re-	
N.		covery of.....	8, 9
New Mexico, run-of-mine coal from, tests of..	13	production of, use of peat in.....	8
New River coal, analyses of.....	16	Sulphur, in producer gas, trouble from.....	47
P.		in gas-producer fuel, percentage of.....	12, 48
Peat, consumption of, per brake horsepower-		T.	
hour.....	24, 26	Tar, commercial use of.....	60-61
relation of, to fuel-bed area.....	24	effect of, on cost of engine lubrication....	46
use of, in production of sulphate of am-		in gas producers, trouble from.....	43-44, 47
monia.....	8	Temperature required at furnace.....	61
Pennsylvania, coal from, analyses of.....	18	Tennessee, run-of-mine coal from, tests of....	13
washery refuse from, tests of.....	13	Texas lignite, analyses of.....	16
Pitch, from gas producers, recovery of.....	9	Turbine. <i>See</i> Steam turbine.	
Pocahontas slack, analyses of.....	15	V.	
Pokings of fuel bed. <i>See</i> Fuel bed.		Vapor, use of, per pound of fuel.....	53
Power, rating of, at producer-gas plants.....	61-62	W.	
<i>See also</i> Horsepower; Producer-gas power.		Washery refuse, tests of.....	13
Preignitions, causes of.....	57-59	Water, in lignite, percentage of.....	12
Producer gas, analyses of.....	28-30	quantity of, for scrubbing gas.....	54-55
applications of.....	62-63	to cool engine.....	55-56
delivery of.....	60	West Virginia, bituminous coal from, analy-	
enrichment of.....	61	ses of.....	18
from anthracite coal, analyses of.....	30	Wood, consumption of, per brake horsepower-	
heating value of.....	30-31	hour.....	26
from bituminous coal, analyses of.....	30	relation of, to fuel-bed area.....	22, 24
heating value of.....	30-31	use of, in producer-gas plants.....	17
production of.....	31	Wood refuse, use of, in gas producers.....	9
from lignite, analyses of.....	30	Wyoming, coal from, tests of.....	13
heating value of.....	30-31		
production of.....	31		



DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

CONCENTRATION EXPERIMENTS WITH THE SILICEOUS  
RED HEMATITE OF THE BIRMINGHAM  
DISTRICT, ALABAMA

BY

JOSEPH T. SINGEWALD, JR.



WASHINGTON  
GOVERNMENT PRINTING OFFICE  
1917

The Bureau of Mines, in carrying out one of the provisions of its organic act—to disseminate information concerning investigations made—prints a limited free edition of each of its publications.

When this edition is exhausted copies may be obtained at cost price only through the Superintendent of Documents, Government Printing Office, Washington, D. C.

The Superintendent of Documents *is not an official of the Bureau of Mines*. His is an entirely separate office, and he should be addressed:

SUPERINTENDENT OF DOCUMENTS,

*Government Printing Office,*

*Washington, D. C.*

The general law under which publications are distributed prohibits the giving of more than one copy of a publication to one person. The price of this publication is 15 cents.

## CONTENTS.

---

	Page.
Introduction.....	7
General field relations and quality of the ores.....	8
Extent of the district.....	8
The ore beds.....	9
Mineralogical composition of the ores.....	9
The problem of concentration.....	10
Earlier work on the beneficiation of the Clinton ores.....	11
Method of conducting the experiments.....	14
Method of plotting results.....	15
Results of concentration tests.....	16
The Big seam.....	16
Upper bench, Big seam.....	17
Tabulated results.....	18
Notes on samples.....	40
Ore from the Big seam.....	40
Sample 1. Ishkooda mine.....	40
Sample 2. Spaulding mine.....	40
Sample 3. Spaulding mine.....	40
Sample 4. Hammond mine.....	42
Sample 5. Ruffner mine.....	43
Sample 6. Ruffner mine.....	43
Comments on tests of upper bench ore.....	45
Big seam ore from Frank White mine.....	46
Sample 7. Frank White mine.....	46
Sample 8. Frank White mine.....	47
Sample 9. Frank White mine.....	47
Sample 10. Frank White mine.....	48
Comments on tests of ore from Frank White mine.....	49
Ore from the lower bench, Big seam.....	50
Sample 11. Ruffner mine.....	50
Sample 12. Spaulding mine.....	52
Sample 13. Spaulding mine.....	53
Sample 14. Spaulding mine.....	53
Sample 15. Spaulding mine.....	55
Sample 16. Spaulding mine.....	55
Sample 17. Spaulding mine.....	56
Comments on tests of lower bench Big seam ore from the Spaulding mine.....	56
Sample 18. Ishkooda mine.....	57
Sample 19. Ishkooda mine.....	57
Sample 20. Ishkooda mine.....	57
Sample 21. Ishkooda mine.....	60
Sample 22. Ishkooda mine.....	60

## Notes on samples—Continued.

	Page.
Comments on tests of ore from lower bench in Ishkooda mine.....	61
Samples from Songo mine.....	61
Sample 23. Songo mine.....	61
Sample 24. Songo mine.....	62
Sample 25. Songo mine.....	62
Sample 26. Songo mine.....	63
Sample 27. Songo mine.....	63
Sample 28. Songo mine.....	64
Comments of tests of ores from Songo mine.....	65
Ore from fossil mine.....	66
Sample 29. Fossil mine.....	66
Sample 30. Fossil mine.....	67
Sample 31. Fossil mine.....	67
Sample 32. Fossil mine.....	67
Sample 33. Fossil mine.....	68
Sample 34. Fossil mine.....	69
Comments on tests of ore from Fossil mine.....	70
Summary of comments on tests of lower bench Big seam ore.....	71
Ore from the Irondale seam.....	72
Sample 35. Ruffner mine.....	72
Sample 36. Ruffner mine.....	73
Sample 37. Hammond mine.....	73
Comments on tests of ore from Irondale seam.....	74
Ore from the Ida seam.....	74
Sample 38. Ruffner mine.....	75
Sample 39. Hammond mine.....	75
Comments on tests of ore.....	75
Résumé of results.....	76
Practical value of the experiments.....	80
Commercial aspects of the problem.....	82
Publications on mineral technology.....	85
Index.....	89

---

ILLUSTRATIONS.

PLATE 1. Topographic map of part of Birmingham District, showing mines on Red Mountain from which samples were taken. In folder.

FIGURE		Page.
1.	Results of tests of sample 1.....	40
2.	Results of tests of sample 2.....	41
3.	Results of tests of sample 3.....	42
4.	Results of tests of sample 4.....	42
5.	Results of tests of sample 5.....	43
6.	Results of tests of sample 5 under 35 mesh.....	44
7.	Results of tests of sample 6.....	44
8.	Results of tests of sample 6 under 35 mesh.....	45
9.	Results of tests of sample 7.....	47
10.	Results of tests of sample 8.....	47
11.	Results of tests of sample 9.....	48
12.	Results of tests of sample 10.....	48



	Page.
FIGURE 13. Results of tests of sample 10 under 35 mesh.....	49
14. Results of tests of sample 11.....	51
15. Results of tests of sample 11 under 35 mesh.....	51
16. Results of tests of sample 12.....	53
17. Results of tests of sample 13.....	53
18. Results of tests of sample 14.....	54
19. Results of tests of sample 14 under 35 mesh.....	54
20. Results of tests of sample 15.....	55
21. Results of tests of sample 16.....	55
22. Results of tests of sample 17.....	56
23. Results of tests of sample 18.....	57
24. Results of tests of sample 19.....	58
25. Results of tests of sample 20.....	59
26. Results of tests of sample 20 under 35 mesh.....	59
27. Results of tests of sample 21.....	60
28. Results of tests of sample 22.....	60
29. Results of tests of sample 23.....	62
30. Results of tests of sample 24.....	62
31. Results of tests of sample 25.....	63
32. Results of tests of sample 26.....	63
33. Results of tests of sample 27.....	64
34. Results of tests of sample 27 under 35 mesh.....	64
35. Results of tests of sample 28.....	65
36. Results of tests of sample 29.....	66
37. Results of tests of sample 29 under 35 mesh.....	67
38. Results of tests of sample 30.....	68
39. Results of tests of sample 31.....	68
40. Results of tests of sample 32.....	69
41. Results of tests of sample 33.....	70
42. Results of tests of sample 34.....	70
43. Results of tests of sample 35.....	73
44. Results of tests of sample 36.....	73
45. Results of tests of sample 37.....	74
46. Results of tests of sample 38.....	75
47. Results of tests of sample 39.....	75



# CONCENTRATION EXPERIMENTS WITH THE SILICEOUS RED HEMATITE OF THE BIRMINGHAM DISTRICT, ALA.

---

By JOSEPH T. SINGEWALD, Jr.

---

## INTRODUCTION.

The possible value of the red hematite iron ore of the southern Appalachian States, if some practicable method of concentrating it could be devised; has long been recognized. The ore is very low grade, and although in enormous quantities, particularly in the States of Alabama, Tennessee, and Georgia, is only at a few points rich enough to work, and even the best of it ranks among the lowest grade iron ores being mined in the United States. A cheap method of beneficiating this ore would make enormous quantities available and vastly increase the iron-ore resources of the country. However, this low-grade ore in effect constitutes a potential iron reserve, and as long as ample quantities of ore can be obtained elsewhere the question of its utilization does not demand immediate notice.

Where these ores are in part of workable grade, however, a problem of more immediate importance arises which, from considerations of economy and prevention of waste in their mining, ought to be solved or, at least, attacked as quickly as possible. Where these ores are being worked they are usually intimately associated with lower-grade material that in mining is left behind. The difficulty and expense of going into ground broken by old workings makes the recovery of this lower-grade ore practically impossible and it is lost forever. Success in concentrating the red ores, therefore, would save material that is now being irretrievably lost. The successful large-scale concentration of low-grade ore in the last few years on the Lake Superior iron ranges should encourage the attack of similar problems in the Birmingham district.

While studying iron-mining methods in the United States for the Bureau of Mines, Dwight E. Woodbridge, consulting mining engineer of the bureau, was impressed with the importance of this problem, and believed it merited the attention of the bureau. Accordingly, Mr. Woodbridge had Mr. W. J. Penhalleton, general superintendent of ore mines and quarries of the Republic Iron & Steel Co., of Birmingham, send the author of this report five samples

of the unworkable siliceous ores of the district. The results of concentration experiments with these ores made a more thorough and systematic investigation desirable. In consequence, the author subsequently collected samples in the district, and these became the basis of the experimental work described in this report. The results are published by the Bureau of Mines as a part of its efforts to increase efficiency in the utilization of mineral resources.

The experiments were conducted in the geological laboratory of the Johns Hopkins University, Baltimore, Md., with the aid of Raymond Leibensperger, mining assistant. The chemical analyses were made by A. C. Fieldner, chemist, of the Bureau of Mines.

The author wishes to express his appreciation of the courtesies extended to him by the mining companies of the district, and particularly to Mr. W. J. Penhallegon, of the Republic Iron & Steel Co. Thanks are also due Prof. W. B. Clark, director of the geological laboratory of the Johns Hopkins University, for the unrestricted use of the facilities of the laboratory.

#### GENERAL FIELD RELATIONS AND QUALITY OF THE ORES.

The general field relations of the ores of the Birmingham district have been fully described by Burchard and Butts,<sup>a</sup> and somewhat the same ground, with greater emphasis on the industrial conditions of the district, is covered by Phillips.<sup>b</sup> For detailed particulars on these points the reader is referred to these two reports, as no attempt will be made here to duplicate the information they present. For the benefit of those not especially interested in these points, and to make the purpose of this report more intelligible to such to whom the above publications are not available, a summary of the general field relations of the district is given.

#### EXTENT OF THE DISTRICT.

The Birmingham district is a narrow belt about 75 miles long and 10 miles wide, with a northeast trend. Near the middle of the belt is the city of Birmingham, which stands in a broad anticlinal valley, known as Birmingham Valley, that extends the whole length of the belt. The iron ores lie in the ridges on both sides of the valley, but by far the most important beds are along the east side, in what is known as Red Mountain. Nearly all of the mining in the district has been confined to that part of Red Mountain lying between Morrow Gap on the northeast and Sparks Gap on the southwest, a dis-

---

<sup>a</sup> Burchard, E. F., and Butts, Charles, Iron ores, fuels, and fluxes of the Birmingham district, Ala., with chapters on the origin of the ores, by E. C. Eckel: Bull. 400, U. S. Geol. Survey, 1910, 204 pp.

<sup>b</sup> Phillips, W. B., Iron making in Alabama: 3d ed., Geol. Survey of Alabama, 1912, 254 pp.



tance of about 25 miles. This part of the district and the location of the mines from which the samples were collected are shown in Plate I. The area of greatest activity is even smaller, and in the summer of 1914, when the iron industry was at a low ebb, mining was practically confined to that part of Red Mountain between Graces Gap and Sparks Gap, a distance of about 10 miles. All the largest mines of the district are included in this latter area.

#### THE ORE BEDS.

The ores are bedded in the Clinton or Rockwood formation, of Silurian age, which outcrops along Red Mountain, and gives the mountain its name. The beds dip away from the valley at angles ranging from  $15^{\circ}$  to  $45^{\circ}$ , but averaging less than  $30^{\circ}$ . One or more ore beds of greater or less importance are everywhere present in the Rockwood formation, and within that part of Red Mountain of greatest economic importance—that is, between Morrow Gap and Sparks Gap—there are four important beds, known, respectively, in their order from top to bottom, as the Hickory Nut, Ida, Big, and Irondale seams. The Hickory Nut seam is the least important of the four and has received little attention. It comprises 3 to 5 feet of sandy ore or ferruginous sandstone, characterized by abundant fossils of the brachiopod *Pentamerus oblongus*, which, on account of looking like a hickory nut in its hull, has given the bed the name. The bed lies about 12 to 20 feet above the Ida seam and reaches its greatest thickness between Birmingham and Bessemer. The other three seams are described on subsequent pages in connection with the results of the concentration experiments.

#### MINERALOGICAL COMPOSITION OF THE ORES.

Mineralogically the ores consist of red hematite and include two structural varieties, oölitic and fossil, which usually occur mixed. Usually the nuclei of the oölitic are small grains of sand, about which successive layers of iron oxide, and frequently thin layers of silica and aluminous material, have been deposited. The fossil ore consists of fragments of such fossil forms as bryozoans, crinoids, corals, and brachiopods.

In addition to the iron oxide the ores carry varying percentages of calcium carbonate, silica, alumina, magnesium carbonate, and other constituents in minor amounts. Where surface waters have leached out the calcium carbonate the ore is porous and friable, and is known as "soft ore"; in contrast, the unaltered ore is called "hard ore." The "hard ores" naturally form the great bulk of the deposits, and those mined at present range in composition from 32 to 45 per cent iron, 5 to 20 per cent lime, 2 to 25 per cent silica, 2 to 5 per cent

alumina, 1 to 3 per cent magnesia, 0.25 to 1.5 per cent phosphorus, from a trace up to 0.5 per cent sulphur, and 0.5 to 3 per cent water.<sup>a</sup> Actually the ores average well under 40 per cent iron, and their value depends chiefly on the relative proportions of lime and silica. Roughly speaking, a self-fluxing ore is one in which the lime is slightly in excess of the silica. As such an ore requires in smelting no addition of limestone, it is equivalent to a higher grade siliceous ore that requires added flux. If the proportion of lime is in excess of that required for a self-fluxing ore, a quantity of siliceous ore can be added. This is done in the district for the most part by using a mixed charge of limy ore and brown ore.

#### THE PROBLEM OF CONCENTRATION.

The great bulk of the red ore of the district is high in silica and comparatively low in lime, so that the addition of considerable flux would be necessary in smelting. The iron content of the ore is so low, however, that the iron content of the furnace charge would be reduced enough to make the use of the ores unfeasible. Only those ores, therefore, that are self-fluxing or nearly self-fluxing can be used at present.

How far the ore can depart from the self-fluxing type and still be workable depends entirely on the status of the iron industry. For example, the upper bench of the Big seam at the Spaulding mine is workable when the demand for iron is good and unworkable when the demand slumps. The average composition of this ore is about 35 to 36 per cent iron, 17 to 18 per cent silica, and 12 per cent lime, and it may be regarded as representing the dividing line between ore that is workable and ore that is not workable under average conditions.

Most of the ore of this composition, or better, lies between Birmingham and Bessemer, and is Big seam ore. But, as is explained more fully in the section dealing with the Big seam ores, only the upper part of the Big seam comes within these limits, consequently the lower part is left in the mines. As the old workings cave, the collapsed roof will make unprofitable the future mining of the rest of the bed. It is with reference to the lower bench of the Big seam in this part of the Birmingham district that this investigation is of most importance. For each ton of ore removed in mining at least one ton of the siliceous lower bench is left in the ground, and as the annual production is three to four million tons, the annual loss is at least that much. The total of the ore reserves in the lower bench

<sup>a</sup> Burchard, E. F., and Butts, Charles, Iron ores, fuels, and fluxes of the Birmingham district, Alabama, with chapters on the origin of ores, by E. C. Eckels: U. S. Geol. Survey Bull. 400, 1910, p. 27.

of the Big seam within the area under consideration is estimated at 146,000,000 tons.<sup>a</sup>

An early solution of the problem of beneficiating these lower bench ores is, therefore, to be sought in order to stop the waste of resources that at some future time will undoubtedly have value. Also, the solution of this problem would place among the available iron-ore reserves of the country a vast tonnage of low-grade ores in other parts of Alabama, as well as in Tennessee and Georgia, and wherever the Clinton ores occur.

For the purpose of determining the possibility of removing silica and concentrating iron in these ores, 39 samples were taken from various mines lying along Red Mountain between the Ruffner mine on the northeast and the Fossil mines on the southwest; 2 of these samples were from the Ida seam, 3 from the Irondale, 24 represented the lower bench of the Big seam, and the other 10 were from the upper bench; 23 of the 24 lower bench samples were taken along Red Mountain southwest of Graces Gap; that is, in the mines where this ore is being lost.

#### EARLIER WORK ON THE BENEFICIATION OF THE CLINTON ORES.

The problem of beneficiating the Clinton ores has been attractive enough to stimulate numerous attacks in the past. Efforts have been directed toward applying some known method of treatment or toward devising some new method especially adapted to these ores. The fact that none of the methods is in use shows that the attempts were not commercially successful. Results of many experiments by private individuals on their own initiative or by private companies have never been made known. Recently an experimental mill, said to have a capacity of about 300 tons in 24 hours, was erected at one of the mines and has made trial runs, but the persons interested refuse to give out any data on the results.

The most extensive series of experiments that have been carried out seem to be those of Phillips,<sup>b</sup> and of Wilkens and Nitze,<sup>c</sup> made nearly 20 years ago.

Results are summarized in Phillips' *Iron Making in Alabama*, chapter IV, concentration of ores. In the first of these experiments the ore was heated to make it magnetic, and then passed over a

<sup>a</sup> Burchard, E. F., and Butts, Charles, *Iron ores, fuels, and fluxes of the Birmingham district, Alabama*, with chapters on the origin of the ores, by E. C. Eckel: U. S. Geol. Survey Bull. 400, 1910, p. 132.

<sup>b</sup> Phillips, W. B., Notes on the magnetization and concentration of iron ores: Trans. Am. Inst. Min. Eng., vol. 25, 1896, pp. 399-423; Concentration of low-grade iron ores: Eng. and Min. Jour., vol. 62, 1896, pp. 75, 76, 105, 106, 124, 125, 151.

<sup>c</sup> Wilkens, H. A. J., and Nitze, H. B. C., The magnetic separation of nonmagnetic material: Trans. Am. Inst. Min. Eng., vol. 26, 1897, pp. 351-370. Discussion of this paper by W. B. Phillips, pp. 1089-1093.



magnetic separator; but the experimenter soon found that magnetic separation could be effected directly. Numerous experiments were made on a working scale and about 500 analyses.

Attention was directed chiefly to the low-grade soft ores, and the experiments demonstrated that by crushing such ores to pass a 15-mesh screen, it was entirely feasible, with a ratio of concentration of 2 to 1 and an iron recovery of about 85 per cent, to make concentrates carrying over 50 per cent iron from ores that in their crude state were worthless. The average composition of eight samples of the low-grade crude tested was 38 per cent iron and 42.5 per cent silica. The average composition of the concentrates obtained was 52.34 per cent iron and 21.63 per cent silica, with a ratio of concentration of 1.93 and an average iron recovery of 71.4 per cent. This average concentrate is of somewhat higher grade than the average soft ore now mined. Phillips estimates the cost of producing a ton of 50 per cent iron concentrates under favorable conditions at about \$1.35; he says that this kind of ore is worth \$1.60 to \$1.80 per ton, and that the price for 55 per cent concentrates would be \$2 per ton.

The results of four tests of samples of hard ore, as given by Phillips, are as follows:

*Results of 4 concentration tests of samples of hard ore.*

Sample.	Crude ore.			Concentrates.			Ratio of concentration.	Iron recovery.
	Iron.	Lime.	Insoluble.	Iron.	Lime.	Insoluble.		
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>		<i>Per cent.</i>
A.....	37.60	15.00	16.20	48.70	9.76	10.26	1.82	71.2
B.....	34.50	17.10	18.04	45.40	11.45	12.25	1.56	84.2
C.....	31.80	10.79	33.10	43.15	8.80	19.66	2.27	59.7
D.....	32.80	9.90	33.70	44.50	9.00	17.30	1.73	78.7
Average.....	34.18	13.20	25.26	45.44	9.75	14.87	1.85	73.5

The first two of these samples represent an almost self-fluxing ore similar to those of the upper bench of the Big seam ores, and although beneficiation has yielded a much higher grade product, ores of this composition can be profitably smelted crude when the demand for iron is strong. Such ores are of slightly higher grade than the upper bench ores of the Spaulding mine, which are just about at the lower workable limit. The last two samples represent ores that more nearly approach the composition of the Ruffner ores described in this report and were decidedly low grade. The results Phillips obtained with these samples were not encouraging, and in this respect are like those the author obtained with the Ruffner ores. Although the concentrate was better than the crude ore, it was not of high enough grade to warrant the cost of treatment.



As the samples tested by Phillips represent two groups of ores of entirely different types, the average of the four samples has no particular significance. It is noteworthy that beneficiation did not reduce the lime content as much as the silica content, so that the concentrates are more nearly self-fluxing than is the crude ore. This result is in accordance with those obtained by the author of this report.

Another point on which the two series of experiments—by Phillips and the author—agree is that the iron content of the concentrates obtained from the soft ores is higher than 50 per cent, whereas that obtained from the hard ores runs below 50 per cent and is usually about 45 per cent.

Unfortunately these experiments throw little light on the amenability to treatment of the siliceous hematites having the composition of those in the lower bench of the Big seam. The upper bench self-fluxing ores are now being worked, and the concentration of the lower bench ores is of immediate importance.

Of great interest, because of the method used being essentially that used in this investigation, are some experiments made in a private laboratory at Wilmington, Del., with what is known as the Moxham-du Pont haloid process. The general results of these experiments have been cited by Burchard <sup>a</sup> and are given here in more detail with the consent of A. J. Moxham. The ore tested was from Pleasant Hill, Ala., and was too low grade for profitable mining. It was ground to pass through a 100-mesh screen and subjected to float and sink tests in troughs containing haloid solutions of specific gravity high enough to float the tailings. Three tests of this ore in solutions of different specific gravity gave the results shown below.

*Results of 3 concentration tests.*

No. of test.	Composition of—						Separation.		Ratio of concentration.	Iron recovered.	Specific gravity of solution.
	Crude ore.		Concentrates.		Tailings.		Concentrates.	Tailings.			
	Iron.	Insoluble.	Iron.	Insoluble.	Iron.	Insoluble.					
A.....	<i>P. ct.</i> 34.32	<i>P. ct.</i> 44.80	<i>P. ct.</i> 53.32	<i>P. ct.</i> 16.90	<i>P. ct.</i> 17.55	<i>P. ct.</i> 72.90	<i>P. ct.</i> 48.8	<i>P. ct.</i> 51.2	2.05	<i>P. ct.</i> 75.7	3.64
B.....	34.32	44.80	42.40	36.28	14.30	78.60	70.8	29.2	1.41	87.4	3.34
C.....	34.32	44.80	41.10	34.50	6.20	90.76	81.2	18.8	1.23	97.2	3.06

From its composition, this ore probably was a low-grade soft ore, and hence the results can not be compared with those of tests of the

<sup>a</sup> Burchard, E. F., in Contributions to economic geology, Preliminary report on the red iron ores of east Tennessee, northeast Alabama, and northwest Georgia: U. S. Geol. Survey Bull. 540, 1914, pp. 325–326; Geol. Survey of Tenn., Bull. 16, 1913, p. 161.

siliceous hard ores. The grade of the first concentrate is better than that of the best soft ores, so that the experiments show the possibility of making an easily marketable concentrate from an ore as siliceous as that used and of saving three-fourths of the iron in the crude ore. The cost of crushing to 100 mesh would probably be prohibitive in commercial work. Beneficiation of other low-grade iron ores by this method is being studied, and an experimental plant of commercial size has been erected for testing the Oriskony limonites at the Rich Patch mines in Virginia. If the method finally proves commercially practicable, it will be of great importance in the treatment of siliceous red hematites.

Ordinary wet methods of concentration are being used for treating low-grade iron ores on the Lake Superior ranges, and have proved so successful that the tonnage treated is continually increasing. However, these Lake Superior ores differ from the siliceous red hematites under consideration, and hence the results obtained can not be applied to the treatment of Clinton ores. Yet the work has shown that the concentration of iron ores by wet methods is feasible on a large scale.

#### METHOD OF CONDUCTING THE EXPERIMENTS.

About a pound of each sample was broken in a small Blake crusher and then crushed in a mortar until it all passed through a 20-mesh screen. A small quantity of the crushed ore was taken as a crude ore sample and the rest was subjected to a screen analysis. For this analysis the Tyler standard screen scale sieves were used, the sizes ranging from 20 mesh to 200 mesh. In these screens the openings are so arranged that they increase in the ratio of the square root of 2, the sizes in the screens used are given in the following table:

*Mesh dimension of screens.*

Mesh.	Opening in—	
	Inches.	Millimeters.
20	0.0328	0.833
28	.0232	.589
35	.0164	.417
48	.0116	.295
65	.0082	.208
100	.0058	.147
150	.0041	.104
200	.0029	.074

As all the sample was finer than 20 mesh, there were eight sizes of material. The seven coarsest were subjected to concentration, but what was finer than 200 mesh was regarded as slimes and was not treated.

To separate the heavier particles, high in iron, from the lighter particles, low in iron and high in silica, a heavy solution, Thoulet solution of a specific gravity of 3.0, was used. Each size of material was treated separately. It was mixed with some of the liquid in a separating funnel; the heavy particles settled to the bottom and the light particles floated, the two portions being then drawn off separately. In this way each sample yielded seven sizes of concentrates and of tailings, besides the slime.

#### METHOD OF PLOTTING RESULTS.

A graphic representation of the results was prepared by plotting the screen analysis as a broken curve, the sizes being used as abscissas and the percentages of the crushed ore as ordinates. The percentages of concentrates and tailings that each size yielded were also shown graphically. A second curve was plotted with the actual quantity of concentrate produced from each mesh size (percentage of concentrate of each size  $\times$  percentage of total ore) as ordinates. This second curve clearly shows how much each mesh size contributes to the total concentrates. As the sum of these products gives the total percentage of concentrates from the sample, and as the slimes are known, the percentage of tailings is obtained by subtracting their sum (concentrates + slimes) from 100. These three percentages are represented by column A (see figs. 1-47) the lower part representing the concentrates, the middle lined part, the slimes, and the upper black part, the tailings. Column B in the figures is likewise divided into three corresponding parts, which show the percentage of the total iron in each. From these figures the ratio of concentration can be calculated for the concentrates alone, or of the product obtained by mixing the slimes and concentrates and considering it as the final product of the treatment.

From all but six samples the concentrates were mixed to form a sample of total concentrates, the tailings were mixed to form a total tailings sample, and these two products were analyzed for iron and silica. Also the crude ore sample and the slimes were analyzed. From these analyses and the percentages of concentrates, tailings, and slimes, the percentage of iron recovered in the concentrates can be calculated and also that of the concentrates plus slimes. As the slimes were of about the same composition as the concentrates, adding them to the concentrates much increased the apparent recovery.

In order to show how iron and silica are distributed in the different mesh sizes, and in the concentrates and tailings obtained from them, the concentrates and tailings from each mesh size of six samples were analyzed separately; and the composition of the crude ore of that mesh size, as well as the composition of the total concentrates



and tailings, were calculated from the analyses. Lime was determined often enough to show the relation of the lime in the crude ore to that in the concentrates. The pages that follow present the detailed results of these experiments.

### RESULTS OF CONCENTRATION TESTS.

#### THE BIG SEAM.

The Big seam, as its name implies, is the important ore bed of the district, and in June, 1914, was the only one being worked. Its general characteristics are described by Burchard,<sup>a</sup> as follows:

The thickness of the Big seam varies from 16 to 30 feet. It extends as a traceable unit on Red Mountain for practically the whole length of the mining district. Notwithstanding the great thickness there are rarely more than 10 to 12 feet of good ore in a single bench, and at most places only 7 to 10 feet are mined. Probably the maximum thickness of the bed, without reference to the thickness of the workable part, occurs between Red Gap (near Irondale) and Bald Eagle, although for a mile southwest of Red Gap the bed remains nearly as thick. From northeast to southwest the total thickness of the ore-bearing sediments gradually decreases, without, however, altering greatly the thickness of the workable portion. About the middle of the district the bed becomes separated into two benches, either by a well-defined parting along the bedding plane, or by a shale bed, thin at first, but thickening gradually to the southwest. The middle of the Big seam is the workable part in the northeast end of the district, but the upper bench is of most importance throughout the rest of the area. In the southwest portion of the district the lower bench, which farther northeast is composed of ore that may eventually be mined, becomes a series of strata of lean ore and shale and is consequently of no possible value; and finally the upper bench itself becomes shaly and carries only a very low-grade ore.

From this description of the Big seam it is evident that mining is removing 50 per cent or less of the ore, and the rest is being left in the ground, where the caving of the workings will make future recovery costly if not impossible. In the part of the district now most productive—that is, along Red Mountain from about south of Birmingham to Bessemer—the siliceous ores form the lower part of the seam and the less siliceous and more highly calcareous ores the upper part. Burchard says that in the northeast part of this strip the two benches are separated by a mere parting along the bedding and in many places there is no parting, the dividing line being purely arbitrary; whereas toward the southwest a thin shale parting that gradually increases in thickness comes in. In this part of the district, then, the upper part of the Big seam is being worked and the lower part left in the ground. Consequently most attention was paid to the lower bench ores of this area. However, several samples of upper bench ores were also tested for com-

---

<sup>a</sup> Burchard, E. F., and Butts, Charles, Iron ores, fuels, and fluxes of the Birmingham district, Alabama, with chapters on the origin of the ores, by E. C. Eckel: U. S. Geol. Survey Bull. 400, 1910, pp. 46, 47.



parison. On the following pages the tests of upper bench ores are considered apart from those of lower bench ores. Several tests were made of ores not included in either of these groups.

## UPPER BENCH, BIG SEAM.

The character of the upper bench ore gradually changes north of a point about opposite Bessemer. Around Bessemer the ore contains a little more lime than is needed to flux the silica and alumina, and to such ore a siliceous ore can be added to make a self-fluxing charge. Northeastward the silica content rises and the lime content declines, until near Birmingham the ore becomes decidedly siliceous.

Phillips<sup>a</sup> says that hard red ore of good grade has the following composition:

*Average composition of good hard ore.*

	Per cent.
Water .....	0.50
Metallic iron .....	37.00
Silica .....	13.44
Lime .....	16.20
Alumina .....	3.18
Phosphorus .....	.37
Sulphur .....	.07
Carbonic acid .....	12.24

The term self-fluxing, so he states, is applied to those varieties of the limy ore in which the lime content equals the sum of the silica and alumina contents. Hence the analysis above represents a good self-fluxing ore. As the alumina content is usually small, a few per cent, a self-fluxing ore may roughly be considered one in which the lime content is a few per cent in excess of the silica content.

The ore from the mines about Readers Gap averages about 34 to 36 per cent Fe, 10 to 13 per cent  $\text{SiO}_2$ , about 18 per cent CaO, and nearly 4 per cent  $\text{Al}_2\text{O}_3$ ; hence its lime content is higher than that required in a self-fluxing ore. Near Spring Gap the lime content is such that the ores are just about self-fluxing, and at Graces Gap the ores have a considerable excess of silica. Thus the ore of the Songo mine, just south of Spring Gap, averages about 34 per cent Fe, 12.5 per cent  $\text{SiO}_2$ , and 16 per cent CaO; whereas that from the Spaulding mine, at Graces Gap, averages about 36 per cent Fe, 19 per cent  $\text{SiO}_2$ , 13 per cent CaO, and nearly 3 per cent  $\text{Al}_2\text{O}_3$ . Northeast of Birmingham the ore grows still more siliceous and at the Ruffner mines averages about 32 per cent Fe, 31 per cent  $\text{SiO}_2$ , 9 per cent CaO, and 3 per cent  $\text{Al}_2\text{O}_3$ . Thus elimination of silica from upper bench Big seam ore is of importance except in the southwestern part of the district.

<sup>a</sup> Phillips, W. B., Iron making in Alabama, 3d ed.: Geol. Survey of Alabama, 1912, p. 35.

## TABULATED RESULTS.

The results of the concentration tests of the 39 samples of ore collected from various mines are shown in the tables following:

TABLE 1.—*Results of crushing and separating samples from various mines.*

## SAMPLE 1. ISHKOODA MINE.

Screen analysis.			Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.		Concentrates.	Tailings.	
	Grams.	Per cent.	Per cent.	Per cent.	Per cent.
Through 20, on 28.....	66.3	21.8	64.5	35.5	14.1
28, on 35.....	74.2	24.5	68.1	31.9	16.7
35, on 48.....	54.1	17.8	66.0	34.0	11.7
48, on 65.....	26.2	8.6	57.1	42.9	4.9
65, on 100.....	24.8	8.2	53.6	46.4	4.4
100, on 150.....	18.3	6.0	57.7	42.3	3.5
150, on 200.....	10.5	3.5	60.9	39.1	2.1
200.....	29.2	a 9.6	.....	.....	0.0
	303.6	100.0	.....	.....	57.4
Total concentrates..... per cent..					57.4
Slimes..... do.....					9.6
Tailings..... do.....					33.0
					100.0

## SAMPLE 2. SPAULDING MINE.

Through 20, on 28.....	129.5	28.2	68.9	31.1	19.4
28, on 35.....	105.8	22.8	73.4	26.6	16.7
35, on 48.....	76.3	16.5	75.2	24.8	12.4
48, on 65.....	41.7	9.0	75.0	25.0	6.8
65, on 100.....	33.8	7.3	73.4	26.6	5.4
100, on 150.....	26.0	5.6	75.5	24.5	4.2
150, on 200.....	17.6	3.8	89.9	10.1	3.4
200.....	31.6	a 6.8	.....	.....	0.0
	462.3	100.0	.....	.....	68.3
Total concentrates..... per cent..					68.3
Slimes..... do.....					6.8
Tailings..... do.....					24.9
					100.0

## SAMPLE 3. SPAULDING MINE.

Through 20, on 28.....	119.4	25.6	68.9	31.1	17.6
28, on 35.....	103.9	22.3	73.1	26.9	16.3
35, on 48.....	75.7	16.2	75.2	24.8	12.2
48, on 65.....	36.6	7.8	69.6	30.4	5.4
65, on 100.....	36.7	7.9	70.4	29.6	5.6
100, on 150.....	32.9	7.0	70.6	29.4	4.9
150, on 200.....	18.9	4.0	72.2	27.8	2.9
200.....	43.1	a 9.2	.....	.....	0.0
	467.2	100.0	.....	.....	64.9
Total concentrates..... per cent..					64.9
Slimes..... do.....					9.2
Tailings..... do.....					25.9
					100.0

a Slimes.

TABLE 1.—*Results of crushing and separating samples from various mines—*  
Continued.

## SAMPLE 4. HAMMOND MINE.

Screen analysis.		Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.	Concentrates.	Tailings.	
	Grams. Per cent.	Per cent.	Per cent.	Per cent.
Through 20, on 28.....	107.2 28.3	47.1	52.9	13.3
28, on 35.....	83.1 22.0	53.9	46.1	11.9
35, on 48.....	54.2 14.3	60.9	39.1	8.7
48, on 65.....	28.4 7.5	71.6	28.4	5.4
65, on 100.....	30.2 8.0	77.4	22.6	6.2
100, on 150.....	25.8 6.8	78.2	21.8	5.3
150, on 200.....	15.4 4.1	79.0	21.0	3.2
200.....	34.2 a 9.0	-----	-----	0.0
	378.5 100.0	-----	-----	54.0
Total concentrates.....per cent..				54.0
Slimes.....do....				9.0
Tailings.....do....				37.0
				100.0

## SAMPLE 5. RUFFNER MINE.

Through 20, on 28.....	52.8 12.5	55.3	44.7	6.9
28, on 35.....	56.0 13.3	55.7	44.3	7.4
35, on 48.....	66.4 15.8	50.7	49.3	8.0
48, on 65.....	63.3 15.0	53.7	46.3	8.1
65, on 100.....	78.9 18.8	53.4	46.6	10.0
100, on 150.....	37.1 8.8	67.9	32.1	6.0
150, on 200.....	19.2 4.5	63.4	36.6	2.9
200.....	47.8 a 11.3	-----	-----	0.0
	421.5 100.0	-----	-----	49.3
Total concentrates.....per cent..				49.3
Slimes.....do....				11.3
Tailings.....do....				39.4
				100.0

## SAMPLE 5. RUFFNER MINE (UNDER 35 MESH).

Through 35, on 48.....	39.7 19.5	47.7	52.3	9.3
48, on 65.....	37.0 18.2	52.5	47.5	9.6
65, on 100.....	47.5 23.4	50.9	49.1	11.7
100, on 150.....	23.2 11.4	58.3	41.7	6.7
150, on 200.....	21.2 10.5	62.5	37.5	6.6
200.....	34.6 a 17.0	-----	-----	0.0
	203.2 100.0	-----	-----	43.9
Total concentrates.....per cent..				43.9
Slimes.....do....				17.0
Tailings.....do....				39.1
				100.0

a Slimes.

TABLE 1.—Results of crushing and separating samples from various mines—Continued.

## SAMPLE 6. RUFFNER MINE.

Screen analysis.			Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.		Concentrates.	Tailings.	
	Grams.	Per cent.	Per cent.	Per cent.	Per cent.
Through 20, on 28.....	55.7	13.6	82.3	17.7	11.2
28, on 35.....	53.2	13.0	72.7	27.3	9.5
35, on 48.....	61.2	15.0	70.5	29.5	10.6
48, on 65.....	59.8	14.6	64.1	35.9	9.4
65, on 100.....	78.9	19.3	56.8	43.2	11.0
100, on 150.....	36.2	8.8	64.3	35.7	5.7
150, on 200.....	19.5	4.8	74.6	25.4	3.6
200.....	44.7	a 10.9	.....	.....	0.0
	409.2	100.0	.....	.....	61.0
Total concentrates.....	per cent..				61.0
Slimes.....	do....				10.9
Tailings.....	do....				28.1
					100.0

## SAMPLE 6. RUFFNER MINE (UNDER 35-MESH).

Through 35, on 48.....	46.7	18.5	49.3	50.7	9.1
48, on 65.....	42.8	17.0	48.0	52.0	8.2
65, on 100.....	65.1	25.8	47.2	52.8	12.2
100, on 150.....	30.9	11.9	55.5	44.5	6.6
150, on 200.....	17.1	7.0	69.5	30.5	4.9
200.....	50.0	a 19.8	.....	.....	0.0
	252.6	100.0	.....	.....	41.0
Total concentrates.....	per cent..				41.0
Slimes.....	do....				19.8
Tailings.....	do....				39.2
					100.0

## SAMPLE 7. FRANK WHITE MINE.

Through 20, on 28.....	112.2	26.7	65.7	34.3	17.5
28, on 35.....	91.7	21.8	73.4	26.6	16.0
35, on 48.....	57.7	13.7	75.7	24.3	10.4
48, on 65.....	33.3	7.9	86.1	13.9	6.8
65, on 100.....	35.4	8.4	86.4	13.6	7.3
100, on 150.....	32.2	7.7	90.2	9.8	6.9
150, on 200.....	18.5	4.4	88.5	11.5	3.9
200.....	39.3	a 9.4	.....	.....	0.0
	420.3	100.0	.....	.....	68.8
Total concentrates.....	per cent..				68.8
Slimes.....	do....				9.4
Tailings.....	do....				21.8
					100.0

a Slimes.



TABLE 1.—Results of crushing and separating samples from various mines—  
Continued.

## SAMPLE 8. FRANK WHITE MINE.

Screen analysis.		Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.	Concentrates.	Tailings.	
	<i>Grams.</i> <i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Through 20, on 28.....	110.2    25.7	75.4    24.6		19.4
28, on 35.....	90.1    21.1	76.9    23.1		16.2
35, on 48.....	62.2    14.5	79.4    20.6		11.5
48, on 65.....	34.3    8.0	85.2    14.8		6.8
65, on 100.....	37.3    8.7	88.1    11.9		7.7
100, on 150.....	33.1    7.7	88.7    11.3		6.8
150, on 200.....	18.5    4.3	88.0    12.0		3.8
200.....	42.8 <i>a</i> 10.0			0.0
	428.5    100.0			72.2
Total concentrates.....per cent..				72.2
Slimes.....do....				10.0
Tailings.....do....				17.8
				100.0

## SAMPLE 9. FRANK WHITE MINE.

Through 20, on 28.....	113.7	28.4	50.1	49.9	14.2
28, on 35.....	93.9	23.4	52.5	47.5	12.3
35, on 48.....	49.1	12.3	65.6	34.4	8.1
48, on 65.....	27.6	6.9	76.9	23.1	5.3
65, on 100.....	32.5	8.1	78.6	21.4	6.4
100, on 150.....	28.4	7.1	80.6	19.4	5.7
150, on 200.....	18.6	4.6	78.2	21.8	3.6
200.....	36.9	<i>a</i> 9.2			0.0
	400.7	100.0			55.6
Total concentrates.....per cent..					55.6
Slimes.....do....					9.2
Tailings.....do....					35.2
					100.0

## SAMPLE 10. FRANK WHITE MINE.

Through 20, on 28.....	51.9	12.7	98.4	1.6	12.5
28, on 35.....	55.6	13.6	96.5	3.5	13.1
35, on 48.....	56.2	13.7	92.9	7.1	12.7
48, on 65.....	40.5	10.0	88.8	11.2	8.9
65, on 100.....	65.2	15.9	70.0	30.0	11.1
100, on 150.....	45.5	11.1	76.3	23.7	8.5
150, on 200.....	23.2	5.7	85.6	14.4	4.9
200.....	71.0	<i>a</i> 17.3			0.0
	409.1	100.0			71.7
Total concentrates.....per cent..					71.7
Slimes.....do....					17.3
Tailings.....do....					11.0
					100.0

*a* Slimes.

TABLE 1.—*Results of crushing and separating samples from various mines—Continued.*

## SAMPLE 10. FRANK WHITE MINE (UNDER 35 MESH).

Screen analysis.			Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.		Concentrates.	Tailings.	
	Grams.	Per cent.	Per cent.	Per cent.	Per cent.
Through 35, on 48.....	26.9	19.1	92.7	7.3	17.7
48, on 65.....	21.6	15.1	88.9	11.1	13.4
65, on 100.....	28.2	20.0	69.7	30.3	13.9
100, on 150.....	20.9	14.8	81.4	18.6	12.0
150, on 200.....	10.3	7.3	92.3	7.7	6.7
200.....	33.5	$\alpha$ 23.7	-----	-----	0.0
	141.4	100.0	-----	-----	63.7
Total concentrates.....per cent..					63.7
Slimes.....do....					23.7
Tailings.....do....					12.6
					100.0

## SAMPLE 11. RUFFNER MINE.

Through 20, on 28.....	37.8	10.2	51.0	49.0	5.2
28, on 35.....	37.8	10.2	49.2	50.8	5.0
35, on 48.....	52.7	14.1	49.5	50.5	7.0
48, on 65.....	68.8	18.5	50.8	49.2	9.4
65, on 100.....	79.0	21.2	50.2	49.8	10.6
100, on 150.....	36.3	9.7	59.1	40.9	5.7
150, on 200.....	17.7	4.7	66.3	33.7	3.1
200.....	42.6	$\alpha$ 11.4	-----	-----	0.0
	372.7	100.0	-----	-----	46.0
Total concentrates.....per cent..					46.0
Slimes.....do....					11.4
Tailings.....do....					42.6
					100.0

## SAMPLE 11. RUFFNER MINE (UNDER 35 MESH).

Through 35, on 48.....	35.0	16.5	40.1	59.9	6.6
48, on 65.....	41.3	19.5	44.1	55.9	8.6
65, on 100.....	62.3	29.4	42.6	57.4	12.5
100, on 150.....	25.0	11.8	49.7	50.3	5.9
150, on 200.....	12.2	5.8	67.3	32.7	3.9
200.....	36.1	$\alpha$ 17.0	-----	-----	0.0
	211.9	100.0	-----	-----	37.5
Total concentrates.....per cent..					37.5
Slimes.....do....					17.0
Tailings.....do....					45.5
					100.0

 $\alpha$  Slimes.

TABLE 1.—*Results of crushing and separating samples from various mines—Continued.*

## SAMPLE 12. SPAULDING MINE.

Screen analysis.		Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.	Concentrates.	Tailings.	
	Grams.	Per cent.	Per cent.	Per cent.
Through 20, on 28.....	110.5	26.6	61.8	38.2
28, on 35.....	101.7	24.5	66.4	33.6
35, on 48.....	65.9	15.9	75.4	24.6
48, on 65.....	31.8	7.6	77.5	22.5
65, on 100.....	32.5	7.8	78.3	21.7
100, on 150.....	26.4	6.4	78.3	21.7
150, on 200.....	15.4	3.6	80.4	19.6
200.....	31.6	a 7.6	-----	-----
	415.8	100.0	-----	64.6
Total concentrates.....			per cent..	64.6
Slimes.....			do.....	7.6
Tailings.....			do.....	27.8
				100.0

## SAMPLE 13. SPAULDING MINE.

Through 20, on 28.....	118.7	27.4	61.2	38.8	16.8
28, on 35.....	99.3	22.9	69.0	31.0	15.8
35, on 48.....	65.5	15.1	73.7	26.3	11.1
48, on 65.....	35.0	8.0	78.2	21.8	6.3
65, on 100.....	34.4	7.9	77.1	22.9	6.1
100, on 150.....	28.2	6.5	79.0	21.0	5.1
150, on 200.....	17.0	3.9	82.3	17.7	3.2
200.....	35.8	a 8.3	-----	-----	0.0
	433.9	100.0	-----	-----	64.4
Total concentrates.....			per cent..	64.4	
Slimes.....			do.....	8.3	
Tailings.....			do.....	27.3	
				100.0	

## SAMPLE 14. SPAULDING MINE.

Through 20, on 28.....	101.0	25.4	61.0	39.0	15.5
28, on 35.....	88.2	22.2	68.7	31.3	15.3
35, on 48.....	61.5	15.5	74.0	26.0	11.5
48, on 65.....	34.7	8.7	75.5	24.5	6.6
65, on 100.....	34.1	8.5	75.0	25.0	6.4
100, on 150.....	23.2	5.8	76.0	24.0	4.4
150, on 200.....	18.8	4.7	76.6	23.4	3.6
200.....	36.5	a 9.2	-----	-----	0.0
	398.0	100.0	-----	-----	63.3
Total concentrates.....			per cent..	63.3	
Slimes.....			do.....	9.2	
Tailings.....			do.....	27.5	
				100.0	

a Slimes.

TABLE 1.—*Results of crushing and separating samples from various mines—Continued.*

## SAMPLE 14. SPAULDING MINE (UNDER 35 MESH).

Screen analysis.			Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.		Concentrates.	Tailings.	
	Grams.	Per cent.	Per cent.	Per cent.	Per cent.
Through 35, on 48.....	63.0	28.2	65.1	34.9	18.4
48, on 65.....	45.0	15.6	68.8	31.2	10.7
65, on 100.....	36.6	16.4	66.4	33.6	10.9
100, on 150.....	25.4	11.3	67.5	32.5	7.6
150, on 200.....	15.8	7.1	71.9	28.1	5.1
200.....	47.9	<sup>a</sup> 21.4	-----	-----	0.0
	223.7	100.0	-----	-----	52.7
Total concentrates.....			per cent..		52.7
Slimes.....			do....		21.4
Tailings.....			do....		25.9
					100.0

## SAMPLE 15. SPAULDING MINE.

Through 20, on 28.....	112.4	27.7	66.6	33.4	18.5
28, on 35.....	88.6	21.8	68.4	31.6	14.9
35, on 48.....	61.9	15.2	75.6	24.4	11.5
48, on 65.....	28.5	7.0	76.0	24.0	5.3
65, on 100.....	31.9	7.8	77.4	22.6	6.0
100, on 150.....	26.1	6.4	77.5	22.5	5.0
150, on 200.....	17.0	4.2	75.6	24.4	3.2
200.....	39.1	<sup>a</sup> 9.9	-----	-----	0.0
	405.5	100.0	-----	-----	64.4
Total concentrates.....			per cent..		64.4
Slimes.....			do....		9.9
Tailings.....			do....		25.7
					100.0

## SAMPLE 16. SPAULDING MINE.

Through 20, on 28.....	110.5	24.7	61.7	38.3	15.2
28, on 35.....	99.0	22.1	70.5	29.5	15.6
35, on 48.....	67.7	15.1	74.8	25.2	11.3
48, on 65.....	41.0	9.2	78.9	21.1	7.9
65, on 100.....	40.5	9.0	77.8	22.2	7.0
100, on 150.....	27.6	6.2	76.6	23.4	4.7
150, on 200.....	18.5	4.1	77.2	22.8	3.2
200.....	43.1	<sup>a</sup> 9.6	-----	-----	0.0
	447.9	100.0	-----	-----	64.9
Total concentrates.....			per cent..		64.9
Slimes.....			do....		9.6
Tailings.....			do....		25.5
					100.0

<sup>a</sup> Slimes.



TABLE 1.—*Results of crushing and separating samples from various mines—Continued.*

## SAMPLE 17. SPAULDING MINE.

Screen analysis.		Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.	Concentrates.	Tailings.	
	Grams. Per cent.	Per cent.	Per cent.	Per cent.
Through 20, on 28.....	110.0 24.2	61.1	38.9	14.4
28, on 35.....	99.9 22.0	65.5	34.5	14.2
35, on 48.....	74.2 16.4	74.2	25.8	12.2
48, on 65.....	36.1 8.0	78.1	21.9	6.4
65, on 100.....	39.0 8.6	74.6	25.4	6.2
100, on 150.....	31.8 7.0	73.6	26.4	5.
150, on 200.....	20.0 4.4	77.5	22.5	3.4
200.....	42.5 <sup>a</sup> 9.4	.....	.....	0.0
	453.5 100.0	.....	.....	62.6
Total concentrates.....per cent..				62.6
Slimes.....do....				9.4
Tailings.....do....				28.0
				100.0

## SAMPLE 18. ISHKOODA MINE.

Through 20, on 28.....	87.1	25.7	67.2	32.8	17.3
28, on 35.....	77.4	22.8	71.8	28.2	16.4
35, on 48.....	52.0	15.3	74.5	25.5	11.4
48, on 65.....	31.6	9.3	75.3	24.7	7.0
65, on 100.....	27.7	8.2	77.2	22.8	6.3
100, on 150.....	19.9	5.9	72.6	27.4	4.3
150, on 200.....	11.8	3.4	73.8	26.2	2.5
200.....	31.9 <sup>a</sup> 9.4	.....	.....	.....	0.0
	339.4 100.0	.....	.....	.....	65.2
Total concentrates.....per cent..					65.2
Slimes.....do....					9.4
Tailings.....do....					25.4
					100.0

## SAMPLE 19. ISHKOODA MINE.

Through 20, on 28.....	100.8	27.2	74.6	25.4	20.3
28, on 35.....	76.5	20.7	80.6	19.4	16.7
35, on 48.....	55.9	15.1	81.3	18.7	12.3
48, on 65.....	31.2	8.4	76.3	23.7	6.4
65, on 100.....	29.8	8.0	77.0	23.0	6.2
100, on 150.....	24.2	6.5	80.0	20.0	5.2
150, on 200.....	14.4	3.9	80.4	19.6	3.1
200.....	37.8 <sup>a</sup> 10.2	.....	.....	.....	0.0
	370.6 100.0	.....	.....	.....	70.2
Total concentrates.....per cent..					70.2
Slimes.....do....					10.2
Tailings.....do....					19.6
					100.0

<sup>a</sup> Slimes.

TABLE 1.—*Results of crushing and separating samples from various mines—Continued.*

## SAMPLE 20. ISHKOODA MINE.

Screen analysis.		Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.	Concentrates.	Tailings.	
	<i>Grams.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Through 20, on 28.....	81.5	28.6	57.8	42.2
28, on 35.....	67.1	23.6	72.8	27.2
35, on 48.....	39.9	14.0	68.9	31.1
48, on 65.....	22.0	7.7	69.0	31.0
65, on 100.....	22.0	7.7	67.1	32.9
100, on 150.....	16.3	5.7	70.3	29.7
150, on 200.....	9.5	3.3	80.2	19.8
200.....	26.8	a 9.4	.....	.....
	285.1	100.0	.....	60.4
Total concentrates.....per cent..				60.4
Slimes.....do....				9.4
Tailings.....do....				30.2
				100.0

## SAMPLE 20. ISHKOODA MINE (UNDER 35 MESH).

Through 35, on 48.....	65.6	28.2	58.5	41.5	16.5
48, on 65.....	36.6	15.7	57.9	42.1	9.1
65, on 100.....	35.7	15.4	60.1	39.9	9.3
100, on 150.....	29.9	12.9	64.3	35.7	8.3
150, on 200.....	15.8	6.8	65.1	34.9	4.4
200.....	48.9	a 21.0	.....	.....	0.0
	232.5	100.0	.....	.....	47.6
Total concentrates.....per cent..					47.6
Slimes.....do....					21.0
Tailings.....do....					31.4
					100.0

## SAMPLE 21. ISHKOODA MINE.

Through 20, on 28.....	48.7	19.7	75.4	24.6	14.9
28, on 35.....	52.1	21.1	80.5	19.5	17.0
35, on 48.....	40.2	16.3	82.1	17.9	13.4
48, on 65.....	22.6	9.1	81.5	18.5	7.4
65, on 100.....	22.0	8.9	79.7	20.3	7.1
100, on 150.....	19.4	7.8	82.8	17.2	6.5
150, on 200.....	11.3	4.6	75.8	24.2	3.5
200.....	30.9	a 12.5	.....	.....	0.0
	247.2	100.0	.....	.....	69.8
Total concentrates.....per cent..					69.8
Slimes.....do....					12.5
Tailings.....do....					17.7
					100.0

a Slimes.

TABLE 1.—Results of crushing and separating samples from various mines—  
Continued.

## SAMPLE 22. ISHKOODA MINE.

Screen analysis.		Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.	Concentrates.	Tailings.	
	<i>Grams.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Through 20, on 28.....	120.2	27.5	70.0	30.0
28, on 35.....	94.9	21.7	77.1	22.9
35, on 48.....	70.6	16.2	81.1	18.9
48, on 65.....	42.3	9.7	78.3	21.7
65, on 100.....	32.8	7.5	76.7	23.3
100, on 150.....	27.7	6.4	77.4	22.6
150, on 200.....	14.5	3.3	82.7	17.3
200.....	33.5	<i>a</i> 7.7	.....	.....
	436.5	100.0	.....	70.1

Total concentrates.....	per cent..	70.1
Slimes.....	do....	7.7
Tailings.....	do....	22.2
		100.0

## SAMPLE 23. SONGO MINE.

Through 20, on 28.....	90.9	25.4	63.5	36.5	16.1
28, on 35.....	79.0	22.1	68.9	31.1	15.1
35, on 48.....	53.9	15.1	72.7	27.3	11.0
48, on 65.....	30.4	8.5	75.8	24.2	6.5
65, on 100.....	31.4	8.8	76.8	23.2	6.8
100, on 150.....	23.8	6.6	76.0	24.0	5.0
150, on 200.....	12.6	3.5	76.0	24.0	2.6
200.....	36.0	<i>a</i> 10.0	.....	.....	0.0
	358.0	100.0	.....	.....	63.1

Total concentrates.....	per cent..	63.1
Slimes.....	do....	10.0
Tailings.....	do....	26.8
		99.9

## SAMPLE 24. SONGO MINE.

Through 20, on 28.....	95.1	25.4	74.6	25.4	18.9
28, on 35.....	75.7	20.2	76.7	23.3	15.5
35, on 48.....	54.9	14.7	78.4	21.6	11.5
48, on 65.....	33.2	8.9	79.3	20.7	7.1
65, on 100.....	34.0	9.1	78.2	21.8	7.1
100, on 150.....	27.7	7.4	75.7	24.3	5.6
150, on 200.....	12.5	3.3	79.4	20.6	2.6
200.....	40.8	<i>a</i> 11.0	.....	.....	0.0
	373.9	100.0	.....	.....	68.3

Total concentrates.....	per cent..	68.3
Slimes.....	do....	11.0
Tailings.....	do....	20.7
		100.0

*a* Slimes.

TABLE 1.—*Results of crushing and separating samples from various mines—Continued.*

## SAMPLE 25. SONGO MINE.

Screen analysis.			Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.		Concentrates.	Tailings.	
	Grams.	Per cent.	Per cent.	Per cent.	Per cent.
Through 20, on 28.....	88.4	22.9	72.8	27.2	16.7
28, on 35.....	77.4	20.0	75.1	24.9	15.0
35, on 48.....	59.5	15.4	78.0	22.0	12.0
48, on 65.....	34.4	8.9	77.7	22.3	6.9
65, on 100.....	35.6	9.2	72.3	27.7	6.6
100, on 150.....	27.5	7.1	77.2	22.8	5.5
150, on 200.....	17.3	4.5	75.7	24.3	3.4
200.....	46.1	$\alpha$ 12.0	.....	.....	0.0
	386.2	100.0	.....	.....	66.1
Total concentrates.....				per cent..	66.1
Slimes.....				do.....	12.0
Tailings.....				do.....	21.9
					100.0

## SAMPLE 26. SONGO MINE.

Through 20, on 28.....	96.3	25.3	67.3	32.7	17.0
28, on 35.....	86.7	22.8	72.5	27.5	16.5
35, on 48.....	57.5	15.1	74.5	25.5	11.2
48, on 65.....	31.6	8.3	77.1	22.9	6.4
65, on 100.....	30.7	8.1	74.5	25.5	6.0
100, on 150.....	25.3	6.6	74.7	25.3	4.9
150, on 200.....	13.1	3.4	80.0	20.0	2.8
200.....	39.6	$\alpha$ 10.4	.....	.....	0.0
	380.8	100.0	.....	.....	64.8
Total concentrates.....				per cent..	64.8
Slimes.....				do.....	10.4
Tailings.....				do.....	24.8
					100.0

## SAMPLE 27. SONGO MINE.

Through 20, on 28.....	109.9	26.2	71.6	28.4	18.8
28, on 35.....	92.5	22.0	75.8	24.2	16.7
35, on 48.....	63.6	15.1	78.3	21.7	11.8
48, on 65.....	36.0	8.6	78.3	21.7	6.7
65, on 100.....	35.8	8.5	78.0	22.0	6.6
100, on 150.....	25.1	6.0	75.2	24.8	4.5
150, on 200.....	15.1	3.6	86.0	14.0	3.1
200.....	42.2	$\alpha$ 10.0	.....	.....	0.0
	420.0	100.0	.....	.....	68.2
Total concentrates.....				per cent..	68.2
Slimes.....				do.....	10.0
Tailings.....				do.....	21.8
					100.0

 $\alpha$  Slimes.



TABLE 1.—*Results of crushing and separating samples from various mines—Continued.*

## SAMPLE 27. SONGO MINE (UNDER 35 MESH).

Screen analysis.			Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.		Concentrates.	Tailings.	
	Grams.	Per cent.	Per cent.	Per cent.	Per cent.
Through 35, on 48.....	72.2	27.8	70.2	29.8	19.5
48, on 65.....	41.8	16.1	71.7	28.3	11.5
65, on 100.....	42.1	16.2	69.0	31.0	11.2
100, on 150.....	30.3	11.6	69.7	30.3	8.1
150, on 200.....	18.7	7.2	76.6	23.4	5.5
200.....	54.9	a 21.1	-----	-----	0.0
	260.0	100.0	-----	-----	55.8
Total concentrates.....				per cent..	55.8
Slimes.....				do....	21.1
Tailings.....				do....	23.1
					100.0

## SAMPLE 28. SONGO MINE.

Through 20, on 28.....	108.2	25.0	71.0	29.0	17.8
28, on 35.....	93.1	21.5	76.0	24.0	16.3
35, on 48.....	70.5	16.3	79.9	20.1	13.0
48, on 65.....	40.5	9.4	80.0	20.0	7.5
65, on 100.....	35.4	8.2	78.4	21.6	6.4
100, on 150.....	26.4	6.1	78.6	21.4	4.8
150, on 200.....	19.0	4.4	80.7	19.3	3.6
200.....	39.4	a 9.1	-----	-----	0.0
	432.5	100.0	-----	-----	69.4
Total concentrates.....				per cent..	69.4
Slimes.....				do....	9.1
Tailings.....				do....	21.5
					100.0

## SAMPLE 29. FOSSIL MINE.

Through 20, on 28.....	93.7	19.0	75.5	24.5	14.3
28, on 35.....	106.4	21.4	81.8	18.2	17.5
35, on 48.....	74.9	15.1	80.6	19.4	12.2
48, on 65.....	49.5	9.9	83.1	16.9	8.2
65, on 100.....	46.1	9.2	80.3	19.7	7.4
100, on 150.....	41.2	8.3	83.8	16.2	7.0
150, on 200.....	29.1	5.8	84.3	15.7	4.9
200.....	56.2	a 11.3	-----	-----	0.0
	497.1	100.0	-----	-----	71.5
Total concentrates.....				per cent..	71.5
Slimes.....				do....	11.3
Tailings.....				do....	17.2
					100.0

a Slimes.

TABLE 1.—*Results of crushing and separating samples from various mines—Continued.*

## SAMPLE 29. FOSSIL MINE (UNDER 35 MESH).

Screen analysis.			Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.		Concentrates.	Tailings.	
	Grams.	Per cent.	Per cent.	Per cent.	Per cent.
Through 35, on 48.....	50.4	25.8	77.6	22.4	20.0
48, on 65.....	31.9	16.4	76.3	23.7	12.5
65, on 100.....	29.6	15.2	73.0	27.0	11.1
100, on 150.....	23.5	12.0	74.9	25.1	9.0
150, on 200.....	15.9	8.1	71.9	28.1	5.8
200.....	43.8	<i>a</i> 22.5	-----	-----	0.0
	195.1	100.0	-----	-----	58.4
Total concentrates.....				per cent..	58.4
Slimes.....				do.....	22.5
Tailings.....				do.....	19.1
					100.0

## SAMPLE 30. FOSSIL MINE.

Through 20, on 28.....	108.3	24.4	69.0	31.0	16.8
28, on 35.....	101.5	22.9	75.9	24.1	17.4
35, on 48.....	70.5	15.9	77.7	22.3	12.4
48, on 65.....	40.1	9.0	77.5	22.5	7.0
65, on 100.....	40.0	9.0	74.4	25.6	6.7
100, on 150.....	29.2	6.6	76.1	23.9	5.0
150, on 200.....	17.2	3.9	76.6	23.4	3.0
200.....	36.8	<i>a</i> 8.3	-----	-----	0.0
	443.6	100.0	-----	-----	68.3
Total concentrates.....				per cent..	68.3
Slimes.....				do.....	8.3
Tailings.....				do.....	23.4
					100.0

## SAMPLE 31. FOSSIL MINE.

Through 20, on 28.....	106.3	25.3	67.3	32.7	17.0
28, on 35.....	92.9	22.1	74.6	25.4	16.5
35, on 48.....	64.2	15.3	77.6	22.4	11.9
48, on 65.....	36.5	8.7	77.8	22.2	6.8
65, on 100.....	36.6	8.7	77.1	22.9	6.7
100, on 150.....	30.3	7.2	75.3	24.7	5.4
150, on 200.....	16.0	3.8	74.7	25.3	2.8
200.....	37.5	<i>a</i> 8.9	-----	-----	0.0
	420.3	100.0	-----	-----	67.1
Total concentrates.....				per cent..	67.1
Slimes.....				do.....	8.9
Tailings.....				do.....	24.0
					100.0

*a* Slimes.

TABLE 1.—*Results of crushing and separating samples from various mines—Continued.*

## SAMPLE 32. FOSSIL MINE.

Screen analysis.			Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.		Concentrates.	Tailings.	
	<i>Grams.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Through 20, on 28.....	109.5	27.4	77.0	23.0	21.1
28, on 35.....	80.0	20.0	81.8	18.2	16.4
35, on 48.....	54.7	13.7	78.4	21.6	10.7
48, on 65.....	33.8	8.5	77.7	22.3	6.6
65, on 100.....	32.8	8.2	76.2	23.8	6.2
100, on 150.....	30.4	7.6	75.6	24.4	5.7
150, on 200.....	19.7	4.9	78.1	21.9	3.8
200.....	38.6	<i>a</i> 9.7	.....	.....	0.0
	399.5	100.0	.....	.....	70.5
Total concentrates.....	per cent..				70.5
Slimes.....	do.....				9.7
Tailings.....	do.....				19.8
					100.0

## SAMPLE 33. FOSSIL MINE.

Through 20, on 28.....	96.8	25.7	69.8	30.2	17.9
28, on 35.....	82.6	21.9	78.1	21.9	17.1
35, on 48.....	58.5	15.5	80.7	19.3	12.5
48, on 65.....	34.6	9.2	80.8	19.3	7.4
65, on 100.....	30.5	8.1	79.1	20.9	6.4
100, on 150.....	25.5	6.7	79.6	20.4	5.3
150, on 200.....	14.2	3.7	77.3	22.7	2.9
200.....	34.8	<i>a</i> 9.2	.....	.....	0.0
	377.5	100.0	.....	.....	69.5
Total concentrates.....	per cent..				69.5
Slimes.....	do.....				9.2
Tailings.....	do.....				21.3
					100.0

## SAMPLE 34. FOSSIL MINE.

Through 20, on 28.....	106.2	25.8	63.8	36.2	16.5
28, on 35.....	88.4	21.4	72.7	27.3	15.6
35, on 48.....	64.3	15.6	75.4	24.6	11.8
48, on 65.....	36.2	8.8	73.9	26.1	6.5
65, on 100.....	33.9	8.2	73.7	26.3	6.0
100, on 150.....	26.9	6.5	74.2	25.8	4.8
150, on 200.....	16.9	4.1	75.9	24.1	3.1
200.....	39.7	<i>a</i> 9.6	.....	.....	0.0
	412.5	100.0	.....	.....	64.3
Total concentrates.....	per cent..				64.3
Slimes.....	do.....				9.6
Tailings.....	do.....				26.1
					100.0

*a* Slimes.

TABLE 1.—Results of crushing and separating samples from various mines—Continued.

## SAMPLE 35. RUFFNER MINE.

Screen analysis.		Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.	Concentrates.	Tailings.	
	<i>Grams.</i> <i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Through 20, on 28.....	129.3    27.1	70.9	29.1	19.2
28, on 35.....	113.4    23.8	78.5	21.5	18.7
35, on 48.....	78.5    16.5	78.2	21.8	12.9
48, on 65.....	35.6    7.5	77.6	22.4	5.8
65, on 100.....	37.7    7.9	77.0	23.0	6.1
100, on 150.....	25.6    5.4	76.7	23.3	4.1
150, on 200.....	19.5    4.0	80.0	20.0	3.2
200.....	37.4 <sup>a</sup> 7.8	.....	.....	0.0
	477.0    100.0	.....	.....	70.0
Total concentrates..... per cent..				70.0
Slimes..... do.....				7.8
Tailings..... do.....				22.2
				100.0

## SAMPLE 36. RUFFNER MINE.

Through 20, on 28.....	95.2	25.3	66.4	33.6	16.8
28, on 35.....	79.2	21.0	74.4	25.6	15.6
35, on 48.....	53.0	14.1	74.7	25.3	10.5
48, on 65.....	34.5	9.2	72.0	28.0	6.6
65, on 100.....	34.7	9.2	70.7	29.3	6.5
100, on 150.....	25.9	6.9	70.5	29.5	4.9
150, on 200.....	16.7	4.4	70.0	30.0	3.1
200.....	37.1	<sup>a</sup> 9.9	.....	.....	0.0
	376.3	100.0	.....	.....	64.0
Total concentrates..... per cent..					64.0
Slimes..... do.....					9.9
Tailings..... do.....					26.1
					100.0

## SAMPLE 37. HAMMOND MINE.

Through 20, on 28.....	103.6	25.8	74.5	25.5	19.2
28, on 35.....	92.4	23.0	83.8	16.2	19.3
35, on 48.....	61.7	15.3	80.4	19.6	12.3
48, on 65.....	32.3	8.0	80.5	19.5	6.4
65, on 100.....	32.6	8.1	82.6	17.4	6.7
100, on 150.....	27.4	6.8	84.7	15.3	5.8
150, on 200.....	14.1	3.5	91.4	8.6	3.2
200.....	38.2	<sup>a</sup> 9.5	.....	.....	0.0
	402.3	100.0	.....	.....	72.9
Total concentrates..... per cent..					72.9
Slimes..... do.....					9.5
Tailings..... do.....					17.6
					100.0

<sup>a</sup> Slimes.



TABLE 1.—*Results of crushing and separating samples from various mines—Continued.*

## SAMPLE 38. RUFFNER MINE.

Screen analysis.			Separation.		Concentrates.
Size of mesh.	Quantity taken for separation.		Concentrates.	Tailings.	
	Grams.	Per cent.	Per cent.	Per cent.	Per cent.
Through 20, on 28.....	103.9	29.7	39.2	60.8	11.6
28, on 35.....	86.1	24.6	42.8	57.2	10.5
35, on 48.....	48.4	13.8	51.6	48.4	7.1
48, on 65.....	24.6	7.0	60.3	39.7	4.2
65, on 100.....	25.9	7.0	65.4	34.6	4.8
100, on 150.....	20.2	5.8	63.7	36.3	3.7
150, on 200.....	11.8	3.4	66.6	33.4	2.3
200.....	29.1	8.3	.....	.....	0.0
	350.0	99.6	.....	.....	44.2
Total concentrates.....	per cent..				44.2
Slimes.....	do....				8.3
Tailings.....	do....				47.5
					100.0

## SAMPLE 39. HAMMOND MINE.

Through 20, on 28.....	81.3	20.7	74.8	25.2	15.5
28, on 35.....	88.1	22.4	66.7	33.3	14.9
35, on 48.....	60.1	15.3	61.0	39.0	9.3
48, on 65.....	33.2	8.7	86.5	13.5	7.5
65, on 100.....	39.7	10.1	89.6	10.4	9.0
100, on 150.....	30.6	7.7	91.3	8.7	7.0
150, on 200.....	16.4	4.1	90.4	9.6	3.7
200.....	43.6	11.0	.....	.....	0.0
	393.0	100.0	.....	.....	66.9
Total concentrates.....	per cent..				66.9
Slimes.....	do....				11.0
Tailings.....	do....				22.1
					100.0

α Slimes.

TABLE 2.—*Composition of products and recoveries of samples from various mines.*

## SAMPLE 1. ISKOOKA MINE.

Products.	Fe.	SiO <sub>2</sub> .	CaO.	Total iron.	Total silica.	Ratio of concentration.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	
Crude.....	32.1	10.3	21.1	100.0	100.0	.....
Concentrates.....	43.5	7.0	12.6	80.7	38.2	1.74
Slimes.....	33.9	7.3	.....	10.5	6.7	.....
Tailings.....	8.2	17.6	.....	8.8	55.1	.....
Concentrates+slimes.....	42.1	7.0	.....	91.2	44.9	1.49

## SAMPLE 2. SPAULDING MINE.

Crude.....	36.9	20.5	11.1	100.0	100.0	.....
Concentrates.....	46.3	10.9	18.5	85.6	34.4	1.46
Slimes.....	44.8	12.1	.....	8.2	3.8	.....
Tailings.....	9.2	49.6	.....	6.2	61.8	.....
Concentrates+slimes.....	46.1	11.0	.....	93.8	38.2	1.33

TABLE 2.—Composition of products and recoveries of samples from various mines—Continued.

## SAMPLE 3. SPAULDING MINE.

Products.	Fe.	SiO <sub>2</sub> .	CaO.	Total iron.	Total silica.	Ratio of concentration.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
Crude.....	35.7	15.0	.....	100.0	100.0	.....
Concentrates.....	46.2	7.9	.....	83.8	34.3	1.54
Slimes.....	40.0	8.9	.....	10.3	5.5	.....
Tailings.....	8.2	34.7	.....	5.9	60.2	.....
Concentrates+slimes.....	45.4	8.0	.....	94.1	39.8	1.35

## SAMPLE 4. HAMMOND MINE.

Crude.....	37.2	40.3	.....	100.0	100.0	.....
Concentrates.....	52.4	17.5	.....	80.7	21.5	1.85
Slimes.....	47.1	23.9	.....	12.1	4.9	.....
Tailings.....	6.8	87.2	.....	7.2	73.6	.....
Concentrates+slimes.....	51.7	18.4	.....	92.8	26.4	1.59

## SAMPLE 5. RUFFNER MINE.

Crude.....	28.8	31.8	10.2	100.0	100.0	.....
Concentrates.....	40.2	20.2	6.9	71.2	30.2	2.07
Slimes.....	40.4	11.6	.....	16.4	4.0	.....
Tailings.....	8.7	55.1	.....	12.4	65.8	.....
Concentrates+slimes.....	40.3	18.6	.....	87.6	34.2	1.65

## SAMPLE 6. RUFFNER MINE.

Crude.....	28.8	29.9	.....	100.0	100.0	.....
Concentrates.....	43.7	16.7	.....	64.0	22.3	2.44
Slimes.....	34.5	13.0	.....	24.4	8.4	.....
Tailings.....	8.3	54.4	.....	11.6	69.3	.....
Concentrates+slimes.....	40.7	15.5	.....	88.4	30.7	1.65

## SAMPLE 7. FRANK WHITE MINE.

Crude.....	45.2	26.1	.....	100.0	100.0	.....
Concentrates.....	53.6	13.4	.....	84.1	32.8	1.45
Slimes.....	53.7	12.7	.....	11.5	4.2	.....
Tailings.....	8.9	81.2	.....	4.4	63.0	.....
Concentrates+slimes.....	53.6	13.3	.....	95.6	37.0	1.28

## SAMPLE 8. FRANK WHITE MINE.

Crude.....	44.0	24.5	.....	100.0	100.0	.....
Concentrates.....	52.3	12.7	.....	84.9	37.8	1.39
Slimes.....	53.0	11.3	.....	11.9	4.6	.....
Tailings.....	8.0	78.6	.....	3.2	57.6	.....
Concentrates+slimes.....	52.4	12.5	.....	96.8	42.4	1.22

## SAMPLE 9. FRANK WHITE MINE.

Crude.....	36.3	41.0	.....	100.0	100.0	.....
Concentrates.....	49.8	19.7	.....	80.3	25.1	1.80
Slimes.....	46.4	23.4	.....	12.4	4.9	.....
Tailings.....	7.2	86.9	.....	7.3	70.0	.....
Concentrates+slimes.....	49.3	20.2	.....	92.7	30.0	1.54

TABLE 2.—Composition of products and recoveries of samples from various mines—Continued.

## SAMPLE 10. FRANK WHITE MINE.

Products.	Fe.	SiO <sub>2</sub> .	CaO.	Total iron.	Total silica.	Ratio of concentration.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
Crude.....	45.8	25.1	-----	100.0	100.0	-----
Concentrates.....	49.8	19.1	-----	69.7	47.2	1.57
Slimes.....	53.7	12.7	-----	28.0	11.7	-----
Tailings.....	8.4	84.3	-----	2.3	41.1	-----
Concentrates+slimes.....	50.9	17.4	-----	97.7	58.9	1.14

## SAMPLE 11. RUFFNER MINE.

Crude.....	27.7	34.3	10.1	100.0	100.0	-----
Concentrates.....	44.0	17.7	4.7	61.0	18.3	2.67
Slimes.....	39.1	11.7	-----	24.6	5.5	-----
Tailings.....	8.6	60.7	-----	14.4	76.2	-----
Concentrates+slimes.....	42.7	15.8	-----	85.6	23.8	1.83

## SAMPLE 12. SPAULDING MINE.

Crude.....	35.9	27.0	-----	100.0	100.0	-----
Concentrates.....	45.5	11.9	-----	83.1	27.7	1.55
Slimes.....	46.1	14.0	-----	9.9	3.8	-----
Tailings.....	8.9	68.4	-----	7.0	68.5	-----
Concentrates+slimes.....	45.6	12.1	-----	93.0	31.5	1.39

## SAMPLE 13. SPAULDING MINE.

Crude.....	36.8	26.2	7.5	100.0	100.0	-----
Concentrates.....	45.6	13.2	7.6	82.1	30.8	1.55
Slimes.....	46.4	13.1	-----	10.8	3.9	-----
Tailings.....	9.3	66.0	-----	7.1	65.3	-----
Concentrates+slimes.....	45.7	13.2	-----	92.9	34.7	1.38

## SAMPLE 14. SPAULDING MINE.

Crude.....	35.3	27.7	-----	100.0	100.0	-----
Concentrates.....	45.1	13.5	-----	81.4	30.4	1.58
Slimes.....	44.3	14.9	-----	11.6	4.8	-----
Tailings.....	8.7	66.3	-----	7.0	64.8	-----
Concentrates+slimes.....	44.9	13.6	-----	93.0	35.2	1.38

## SAMPLE 15. SPAULDING MINE.

Crude.....	36.5	26.1	-----	100.0	100.0	-----
Concentrates.....	45.0	14.0	-----	81.4	32.8	1.55
Slimes.....	44.4	14.4	-----	12.3	5.2	-----
Tailings.....	8.7	66.6	-----	6.3	62.0	-----
Concentrates+slimes.....	44.9	14.1	-----	93.7	38.0	1.35

## SAMPLE 16. SPAULDING MINE.

Crude.....	36.9	26.1	-----	100.0	100.0	-----
Concentrates.....	44.9	13.1	-----	81.2	31.1	1.54
Slimes.....	46.5	13.5	-----	12.4	4.4	-----
Tailings.....	9.0	69.1	-----	6.4	64.5	-----
Concentrates+slimes.....	45.1	13.0	-----	93.6	35.5	1.34

TABLE 2.—Composition of products and recoveries of samples from various mines—Continued.

## SAMPLE 17, SPAULDING MINE.

Products.	Fe.	SiO <sub>2</sub> .	CaO.	Total iron.	Total silica.	Ratio of concentration.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
Crude.....	35.0	27.5	-----	100.0	100.0	-----
Concentrates.....	47.7	12.3	-----	82.3	27.8	1.60
Slimes.....	43.3	15.4	-----	11.2	5.2	-----
Tailings.....	8.4	66.4	-----	6.5	67.0	-----
Concentrates and slimes.....	47.1	12.7	-----	93.5	33.0	1.39

## AVERAGE OF SAMPLES 12 TO 17.

Crude.....	36.1	26.8	7.5	-----	-----	-----
Concentrates.....	45.6	13.0	7.6	81.9	-----	1.56
Slimes.....	45.2	14.2	-----	-----	-----	-----
Tailings.....	8.8	67.1	-----	-----	-----	-----
Concentrates and slimes.....	45.6	13.1	-----	93.3	-----	1.37

## SAMPLE 18, ISHKOODA MINE.

Crude.....	33.5	29.7	8.0	100.0	100.0	-----
Concentrates.....	43.7	14.2	7.7	80.6	33.9	1.55
Slimes.....	45.4	13.2	-----	12.1	4.5	-----
Tailings.....	10.2	66.5	-----	7.3	61.6	-----
Concentrates and slimes.....	43.9	14.0	-----	92.7	38.4	1.34

## SAMPLE 19, ISHKOODA MINE.

Crude.....	39.1	20.5	-----	100.0	100.0	-----
Concentrates.....	47.1	10.6	-----	83.5	39.8	1.42
Slimes.....	45.3	10.6	-----	11.6	5.8	-----
Tailings.....	9.8	52.0	-----	4.9	54.4	-----
Concentrates and slimes.....	46.9	10.6	-----	95.1	45.6	1.24

## SAMPLE 20, ISHKOODA MINE.

Crude.....	32.4	29.3	-----	100.0	100.0	-----
Concentrates.....	42.3	16.1	-----	80.0	34.5	1.66
Slimes.....	39.8	12.4	-----	11.7	4.1	-----
Tailings.....	8.8	57.3	-----	8.3	61.4	-----
Concentrates and slimes.....	42.0	15.6	-----	91.7	38.6	1.44

## SAMPLE 21, ISHKOODA MINE.

Crude.....	42.3	18.2	-----	100.0	100.0	-----
Concentrates.....	48.9	9.6	-----	81.4	37.6	1.44
Slimes.....	49.0	11.2	-----	14.6	7.8	-----
Tailings.....	9.4	55.0	-----	4.0	54.6	-----
Concentrates and slimes.....	48.9	9.8	-----	96.0	45.4	1.21

## SAMPLE 22, ISHKOODA MINE.

Crude.....	38.3	20.2	-----	100.0	100.0	-----
Concentrates.....	46.2	11.3	-----	85.9	37.7	1.43
Slimes.....	45.0	11.6	-----	9.2	4.2	-----
Tailings.....	8.4	55.0	-----	4.9	53.1	-----
Concentrates and slimes.....	46.1	11.3	-----	95.1	41.9	1.30



TABLE 2.—Composition of products and recoveries of samples from various mines—Continued.

## AVERAGE OF SAMPLES 18 TO 22.

Products.	Fe.	SiO <sub>2</sub> .	CaO.	Total iron.	Total silica.	Ratio of concentration.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
Crude.....	37.1	25.6	8.0			
Concentrates.....	45.6	12.4	7.7	82.3		1.50
Slimes.....	44.9	11.8				
Tailings.....	9.3	57.2				
Concentrates and slimes.....	45.6	12.3		94.1		1.31

## SAMPLE 23. SONGO MINE.

Crude.....	37.1	21.8		100.0	100.0	
Concentrates.....	45.4	12.7		81.1	32.0	1.58
Slimes.....	43.9	12.9		12.4	5.1	
Tailings.....	8.6	58.8		6.5	62.9	
Concentrates+slimes.....	45.2	12.7		93.5	37.1	1.37

## SAMPLE 24. SONGO MINE.

Crude.....	38.2	19.7		100.0	100.0	
Concentrates.....	45.3	12.3		82.5	40.4	1.46
Slimes.....	44.1	11.2		12.9	5.9	
Tailings.....	8.4	53.9		4.6	53.7	
Concentrates+slimes.....	45.1	12.1		95.4	46.3	1.26

## SAMPLE 25. SONGO MINE.

Crude.....	39.1	21.9		100.0	100.0	
Concentrates.....	46.2	12.6		78.1	37.9	1.51
Slimes.....	45.8	14.1		14.0	7.7	
Tailings.....	14.1	54.4		7.9	54.4	
Concentrates+slimes.....	46.1	12.8		92.1	45.6	1.28

## SAMPLE 26. SONGO MINE.

Crude.....	39.3	23.2		100.0	100.0	
Concentrates.....	44.9	12.4		81.1	33.6	1.54
Slimes.....	43.7	13.5		12.7	5.8	
Tailings.....	9.0	58.6		6.2	60.6	
Concentrates+slimes.....	44.7	12.6		93.8	39.4	1.33

## SAMPLE 27. SONGO MINE.

Crude.....	37.2	21.1	10.5	100.0	100.0	
Concentrates.....	43.5	13.2	9.5	83.2	37.5	1.47
Slimes.....	44.0	12.8		12.3	5.3	
Tailings.....	7.4	63.0		4.5	57.2	
Concentrates+slimes.....	43.6	13.1		95.5	42.8	1.28

## SAMPLE 28. SONGO MINE.

Crude.....	39.5	19.3		100.0	100.0	
Concentrates.....	45.3	11.8		83.9	38.0	1.44
Slimes.....	47.4	11.7		11.5	4.9	
Tailings.....	8.0	57.2		4.6	57.1	
Concentrates+slimes.....	45.5	11.8		95.4	42.9	1.27

TABLE 2.—Composition of products and recoveries of samples from various mines—Continued.

## AVERAGE OF SAMPLES 23 TO 28.

Products.	Fe.	SiO <sub>2</sub> .	CaO.	Total iron.	Total silica.	Ratio of concentration.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
Crude.....	38.4	21.2	10.5			
Concentrates.....	45.1	12.5	9.5	81.7		1.50
Slimes.....	44.8	12.7				
Tailings.....	9.3	57.7				
Concentrates+slimes.....	45.0	12.5		94.3		1.30

## SAMPLE 29. FOSSIL MINE.

Crude.....	39.8	21.1		100.0	100.0	
Concentrates.....	48.1	11.7		71.1	31.8	1.71
Slimes.....	43.8	16.6		25.0	17.4	
Tailings.....	8.1	57.1		3.9	50.8	
Concentrates+slimes.....	46.9	13.1		96.1	49.2	1.24

## SAMPLE 30. FOSSIL MINE.

Crude.....	37.8	22.0		100.0	100.0	
Concentrates.....	46.0	11.9		84.5	36.2	1.46
Slimes.....	44.6	13.6		9.9	5.0	
Tailings.....	8.9	56.3		5.6	58.8	
Concentrates+slimes.....	45.8	12.1		94.4	41.2	1.31

## SAMPLE 31. FOSSIL MINE.

Crude.....	36.8	21.7	9.6	100.0	100.0	
Concentrates.....	45.3	12.8	7.6	84.0	38.8	1.49
Slimes.....	43.0	13.7		10.6	5.5	
Tailings.....	8.2	51.4		5.4	55.7	
Concentrates+slimes.....	45.0	12.9		94.6	44.3	1.32

## SAMPLE 32. FOSSIL MINE.

Crude.....	39.4	18.6		100.0	100.0	
Concentrates.....	46.2	11.9		84.7	42.4	1.42
Slimes.....	44.2	12.4		11.1	6.1	
Tailings.....	8.3	51.5		4.2	51.5	
Concentrates+slimes.....	46.0	12.0		95.8	48.5	1.35

## SAMPLE 33. FOSSIL MINE.

Crude.....	38.4	20.2		100.0	100.0	
Concentrates.....	45.3	12.0		84.0	38.5	1.44
Slimes.....	45.5	11.3		11.2	4.8	
Tailings.....	8.5	57.5		4.8	56.7	
Concentrates+slimes.....	45.3	11.9		95.2	43.3	1.27

## SAMPLE 34. FOSSIL MINE.

Crude.....	35.6	21.5		100.0	100.0	
Concentrates.....	44.9	12.2		82.4	35.3	1.56
Slimes.....	42.0	12.9		11.5	5.6	
Tailings.....	8.2	50.4		6.1	59.1	
Concentrates+slimes.....	44.5	12.3		93.9	38.9	1.35

TABLE 2.—Composition of products and recoveries of samples from various mines—Continued.

## AVERAGE OF SAMPLES 30 TO 34.

Products.	Fe.	SiO <sub>2</sub> .	CaO.	Total iron.	Total silica.	Ratio of concentration.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
Crude.....	37.6	20.8	9.6	.....	.....	.....
Concentrates.....	45.5	12.2	7.6	83.8	.....	1.47
Slimes.....	43.9	12.8	.....	.....	.....	.....
Tailings.....	8.4	53.3	.....	.....	.....	.....
Concentrates + slimes.....	45.3	12.2	.....	94.8	.....	1.30

## SAMPLE 35. RUFFNER MINE.

Crude.....	38.1	22.1	.....	100.0	100.0	.....
Concentrates.....	45.1	13.7	.....	85.4	40.4	1.43
Slimes.....	44.3	14.2	.....	9.4	4.7	.....
Tailings.....	8.6	58.6	.....	5.2	54.9	.....
Concentrates + slimes.....	44.9	13.8	.....	94.8	45.1	1.29

## SAMPLE 36. RUFFNER MINE.

Crude.....	34.5	23.2	.....	100.0	100.0	.....
Concentrates.....	43.5	15.0	.....	82.8	40.0	1.56
Slimes.....	40.4	13.3	.....	11.9	5.5	.....
Tailings.....	6.9	50.1	.....	5.3	54.5	.....
Concentrates + slimes.....	43.1	14.7	.....	94.7	45.5	1.35

## SAMPLE 37. HAMMOND MINE.

Crude.....	41.5	20.9	.....	100.0	100.0	.....
Concentrates.....	46.8	14.0	.....	84.6	45.3	1.37
Slimes.....	50.0	11.1	.....	11.8	4.7	.....
Tailings.....	8.4	64.1	.....	3.6	50.0	.....
Concentrates + slimes.....	47.2	13.7	.....	96.4	50.0	1.21

## SAMPLE 38. RUFFNER MINE.

Crude.....	28.2	40.8	.....	100.0	100.0	.....
Concentrates.....	47.5	14.7	.....	77.8	14.5	2.26
Slimes.....	38.9	23.8	.....	12.0	4.4	.....
Tailings.....	5.8	76.6	.....	10.2	81.1	.....
Concentrates + slimes.....	46.1	16.1	.....	89.8	18.9	1.90

## SAMPLE 39. HAMMOND MINE.

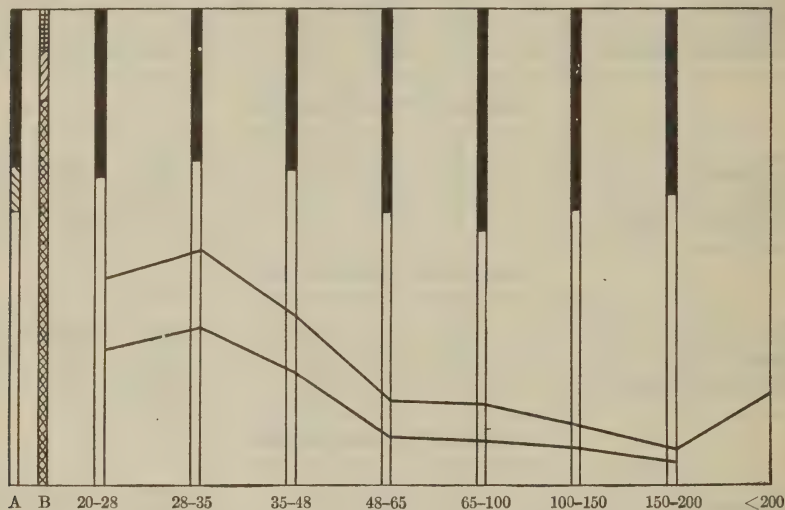
Crude.....	46.2	28.0	.....	100.0	100.0	.....
Concentrates.....	55.2	14.6	.....	82.9	32.1	1.50
Slimes.....	55.5	13.3	.....	13.7	4.8	.....
Tailings.....	6.8	87.1	.....	3.4	63.1	.....
Concentrates + slimes.....	55.3	14.4	.....	96.6	36.9	1.28

## NOTES ON SAMPLES.

## ORE FROM THE BIG SEAM.

## SAMPLE 1. ISHKOODA MINE.

Sample 1 was taken in No. 13 slope, 6 right heading, near the man-way. The upper bench is here underlaid with a  $\frac{1}{2}$ -inch shale parting and is 7 feet thick. The sample was taken across the entire bench.

FIGURE 1.—Results of tests of sample 1.<sup>a</sup>

## SAMPLE 2. SPAULDING MINE.

Sample 2, from 1 left heading, cross left, 700 feet from slope. The Big seam has no parting in this mine, so that the lower limit of the upper bench is an arbitrary line, and the upper 11 feet of the seam were mined when the mine was working. The sample represents the entire 11 feet of workable ore.

## SAMPLE 3. SPAULDING MINE.

Sample 3 was taken from 3 left heading at the breast and represents the upper 12 feet of the seam or the part that was worked at this point.

Instead of the concentrates and tailings for the entire sample being analyzed, the concentrates and tailings for each mesh size were analyzed, and the composition of the concentrates and tailings, as given above, was calculated from these results. The analyses of the

<sup>a</sup> In figs. 1 to 47, column A shows, in upward order, the percentage of concentrate, slime, and tailing from the sample, and column B the percentage in each of the total iron in the sample. Upper broken curve shows proportion of total crushed ore in each size; lower broken curve shows actual quantity of concentrate from each size. See p. 15.



products obtained from the different mesh sizes follow. For each mesh size the percentage of the total iron that went into the con-

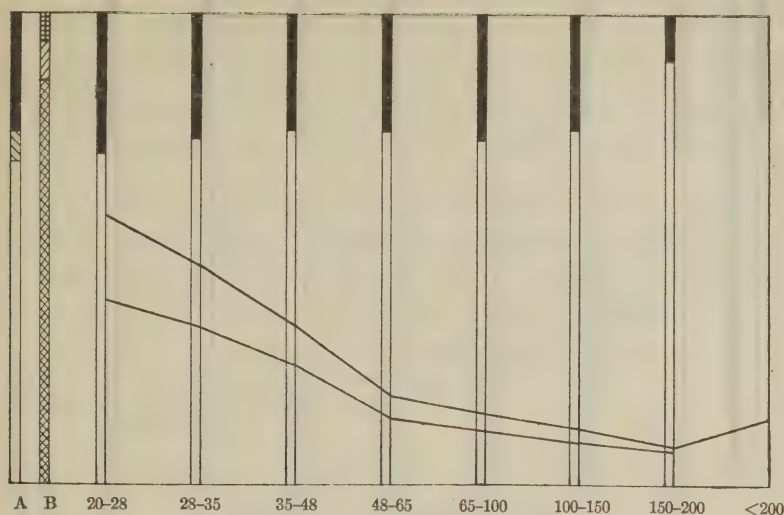


FIGURE 2.—Results of tests of sample 2.

centrates and the tailings in that size is given, and also the composition of the crude ore of each size.

*Composition of concentrates and tailings by mesh sizes.*

Size of mesh.	Concentrates.		Tailings.		Crude.		Total iron in—	
	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Concentrates.	Tailings.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
20 to 28.....	42.9	9.5	9.3	44.7	32.5	20.5	91.1	8.9
28 to 35.....	45.7	8.1	10.0	36.9	36.1	15.9	92.6	7.4
35 to 48.....	46.4	7.6	8.9	33.5	37.1	14.0	94.0	6.0
48 to 65.....	46.8	6.9	8.1	25.0	35.0	12.4	93.0	7.0
65 to 100.....	47.5	6.9	6.4	24.9	35.3	12.2	94.7	5.3
100 to 150.....	51.7	6.1	6.6	22.5	38.4	10.9	95.0	5.0
150 to 200.....	55.1	5.7	9.0	20.0	42.3	9.7	94.1	5.9
200.....					40.0	8.9		

The above table shows that the 20 to 28 mesh crude ore is of decidedly lower grade than the original ore, the 28 to 35 mesh size is of about the same grade, and the smaller sizes are all better than the original, there being a nearly constant increase in grade for each successively smaller size. If at the outset of concentration the 20 to 28 mesh size were rejected and thrown into the tailings, with a ratio of concentration of 2.11 and an iron recovery of 62.8 per cent, one would obtain a concentrate running 47.4 per cent Fe and 7.3

per cent  $\text{SiO}_2$ . In other words, the compensation for a decrease in iron recovery of 21 per cent would be by only an increase of 1.2 per

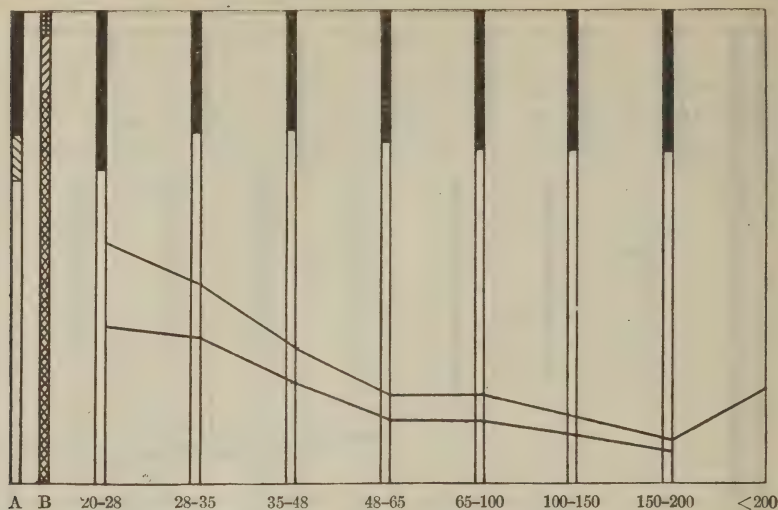


FIGURE 3.—Results of tests of sample 3.

cent iron and a decrease of 0.6 per cent silica in the concentrates, not enough to warrant the procedure.

#### SAMPLE 4. HAMMOND MINE.

The Big seam was not worked at this mine, but it is cut by an adit driven to reach the slope on the Irondale seam, and the sample was

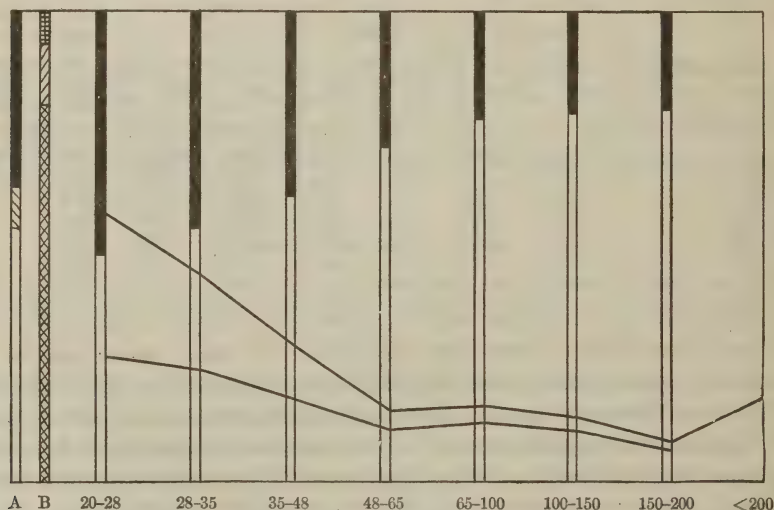


FIGURE 4.—Results of tests of sample 4.

taken from the exposure in this adit. The seam here is only a few feet from the surface, and the ore corresponds more nearly to soft ore. The sample was taken across the entire seam.

## SAMPLE 5. RUFFNER MINE.

The Big seam in the Ruffner mine is about 18 feet thick, the upper 7 feet being workable, and is overlain by what is called "honeycomb" rock, which is really a low-grade ore high in silica with considerable visible calcite. The sample was taken from the 7 feet of workable ore at the bottom of the slope at heading 10 left. The ore is unusually fine grained, as is shown by the screen-analysis curves (figs. 5 to 8).

Because the ore was so fine grained another part of the sample was crushed to pass 35 mesh and subjected to separation. The results of this test, given in Table 1 (p. 19), show a screen-analysis curve (fig. 6) of the same character as that from the 20-mesh test and the

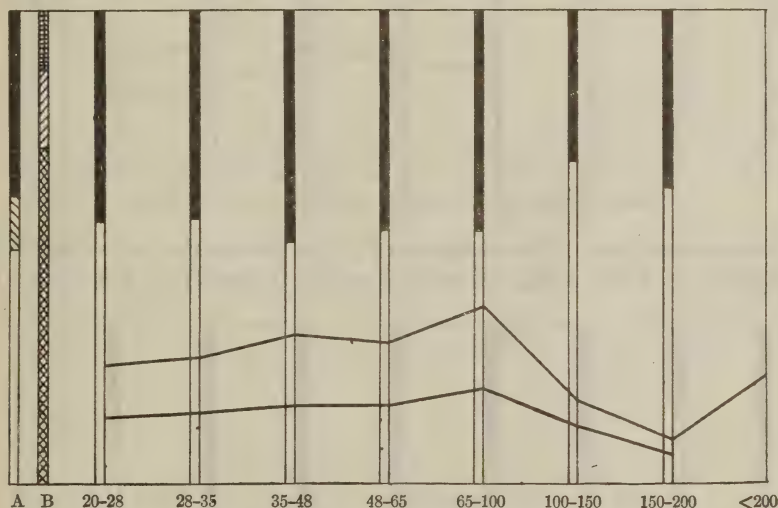


FIGURE 5.—Results of tests of sample 5.

same percentage of tailings in the concentration tests. The only evident change is an increase of slimes and a decrease in the quantity of concentrates with finer crushing. Though no analysis was made of the products, the various percentages seem to indicate that little was gained by the finer crushing.

## SAMPLE 6. RUFFNER MINE.

Sample 6 was taken from 4 left heading, 400 feet from the slope, and represents the upper 7 feet of ore. This ore is very fine grained, as the screen-analysis curves indicate. Only the concentrates from this separation were analyzed. They carried 42.5 per cent Fe and 20.6 per cent  $\text{SiO}_2$ . Another part of this sample was crushed under 35 mesh and concentrated; the results are given on page 20.

Crushing this sample finer than 35 mesh increased the ratio of concentration from 1.64 to 2.44, and correspondingly increased the grade

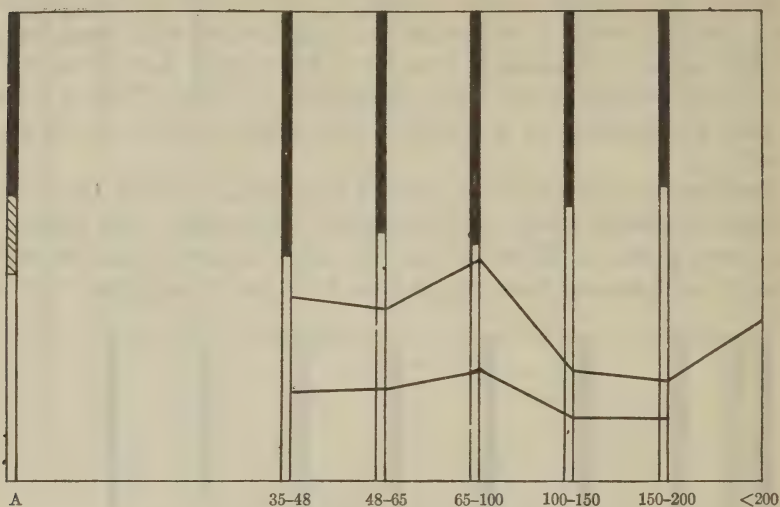


FIGURE 6.—Results of tests of sample 5 under 35 mesh.

of the concentrates. The increase of slimes leaves the advantage still decidedly in favor of the 35-mesh size, for if the slimes are added to

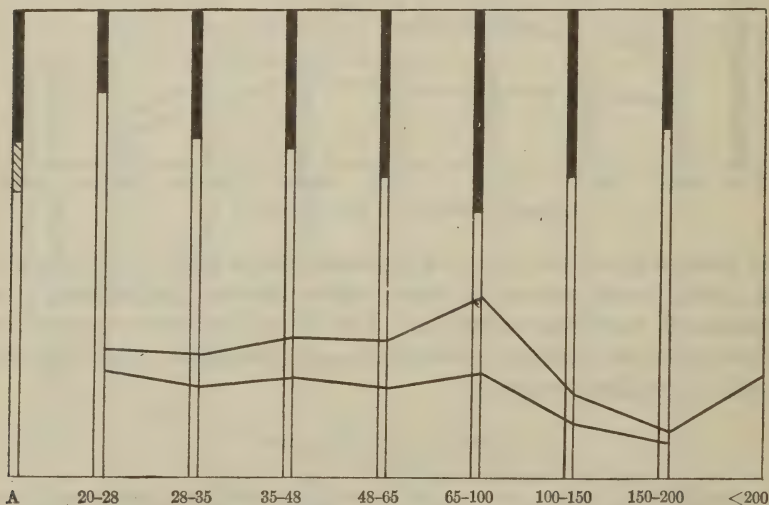


FIGURE 7.—Results of tests of sample 6.

the concentrates the ratio of concentration becomes 1.65, which is practically that of the concentrates in the 20-mesh size, and yet the



composition of the combined product runs only 1.5 per cent less in iron and 5 per cent less in silica. If the slimes are rejected and only the concentrates considered, the finer crushing yields a product 1.2 per cent higher in iron and 3.9 per cent lower in silica, but decreases the iron recovery to 64 per cent.

#### COMMENTS ON TESTS OF UPPER BENCH ORE.

Of the preceding six samples of Big seam ore the Hammond sample represents soft ore from the entire thickness of the seam, the Ishkooda sample is self-fluxing, and the other four are siliceous hard ores. The two from the Spaulding mine are comparatively

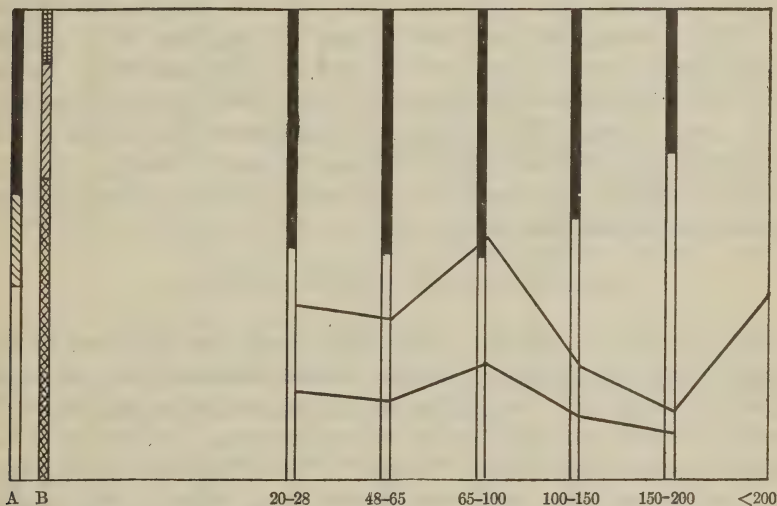


FIGURE 8.—Results of tests of sample 6 under 35 mesh.

good for ores of that type, but the two from the Ruffner mine are low in iron and high in silica and can not be utilized in their present condition.

Concentration of the self-fluxing Ishkooda ore yielded a self-fluxing concentrate more than 10 per cent higher in iron and 3 per cent lower in silica than the original ore. Though the product is more valuable, concentration would hardly be justified because the crude ore is itself marketable.

The Spaulding ores represent those that are too siliceous to work except when the iron market is strong. The average of the two samples is 36.3 per cent Fe and 17.8 per cent  $\text{SiO}_2$ , and in one sample the lime content is 11.1. The concentrates average 46.3 per cent Fe and 9.4 per cent  $\text{SiO}_2$ , and the lime content of one sample is 8.5. In other words, concentration of a decidedly siliceous ore yielded a con-

concentrate carrying 10 per cent more iron, 8.4 per cent less silica, and nearly self-fluxing. Addition of the slimes to the concentrates makes a product averaging 45.8 per cent Fe and 9.5 per cent  $\text{SiO}_2$ , a reduction of only 0.5 per cent Fe and an increase of 0.1  $\text{SiO}_2$ , with an increase of iron recovery from 84.7 to 94 per cent. Thus the concentration of the upper-bench Spaulding ore, which has been utilized only when the iron trade was most active, yielded a product better than the best of the crude ores of the district.

The soft Hammond ore, which in its crude state is too low in iron and high in silica to have any value, yielded a concentrate somewhat better than the average soft ore of the district, which according to Phillips<sup>a</sup> carries 50.80 per cent Fe and 18.50 per cent  $\text{SiO}_2$ .

Least promising are the results from the two samples of Ruffner ore. They average 28.8 per cent Fe and 30.9 per cent  $\text{SiO}_2$ , and the concentrates average 42 per cent Fe and 18.5 per cent  $\text{SiO}_2$ . In one sample the lime content fell from 10.2 in the crude ore to 6.9 per cent in the concentrates. The average of concentrates plus slimes is 40.5 per cent Fe and 17.1 per cent  $\text{SiO}_2$ , and the iron recovery is thereby increased from 67.6 per cent to 88 per cent. Either product is below the limits of marketable ore.

#### BIG SEAM ORE FROM FRANK WHITE MINE.

Four samples of soft ore were taken from the open cut of the Frank White mine, on the summit of Red Mountain, at the south end of Birmingham. An incline leads from this open cut, which is several hundred feet long, to the railroad tracks. At the top of the incline the entire thickness, 18 feet, of the Big seam is exposed. Only the upper 6 feet has been worked, and from it two samples were taken, one from each side of the incline. A third sample was taken at the incline, from the middle 6 feet of the bed, and a fourth at the same place from the lowest 6 feet, which is much finer grained than the rest of the bed, as indicated by the screen-analysis curves (figs. 12 and 13). The ore from the middle 6 feet is similar in appearance to that from the upper, but, as the analyses show, is much poorer, being high in silica and low in iron for a soft ore. Because of the three divisions of the bed, and the samples being taken at the outcrop, they are considered apart from the rest of the Big seam samples.

#### SAMPLE 7. FRANK WHITE MINE.

Sample 7 represents the upper 6 feet of the Big seam in the part of the open cut south of the incline.

<sup>a</sup> Phillips, W. B., *Iron making in Alabama*, 3d ed.: Alabama Geological Survey, 1912, p. 30.

## SAMPLE 8. FRANK WHITE MINE.

Sample 8 represents the upper 6 feet of the Big seam in the part of the open cut north of the incline.

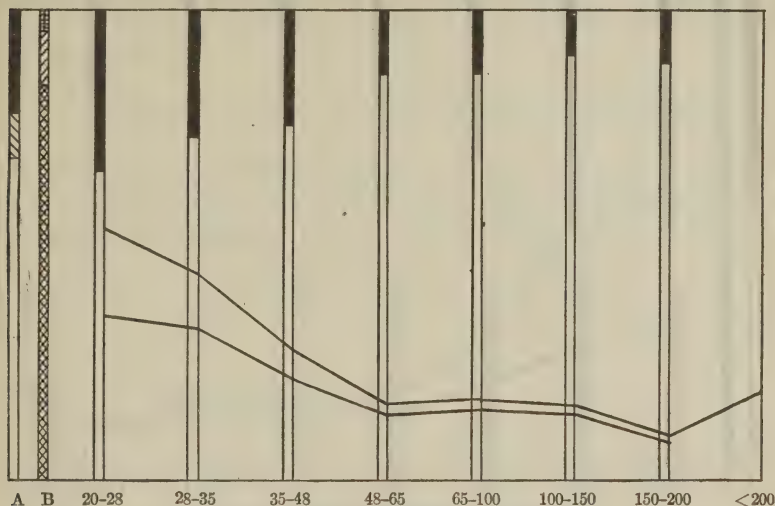


FIGURE 9.—Results of tests of sample 7.

## SAMPLE 9. FRANK WHITE MINE.

Sample 9 was taken from the middle 6 feet of the Big seam, where it is cut by the incline at the top of the ridge.

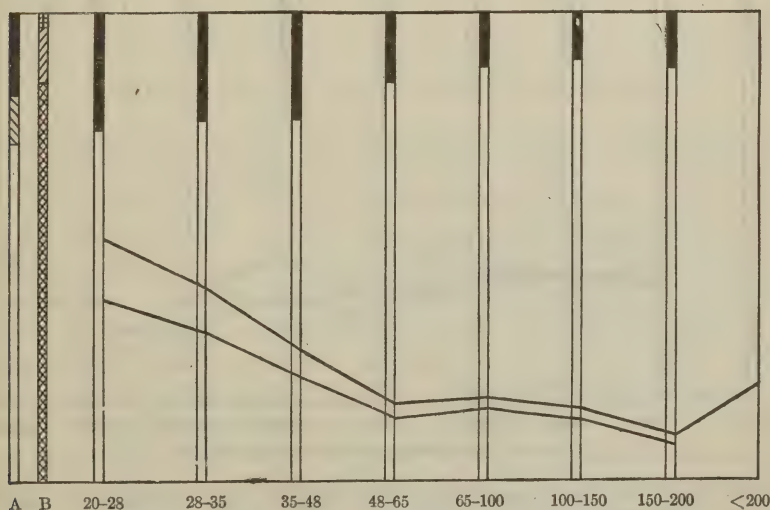


FIGURE 10.—Results of tests of sample 8.

## SAMPLE 10, FRANK WHITE MINE.

Sample 10 was taken from the lowest 6 feet of the Big seam, where it is cut by the incline at the top of the ridge.

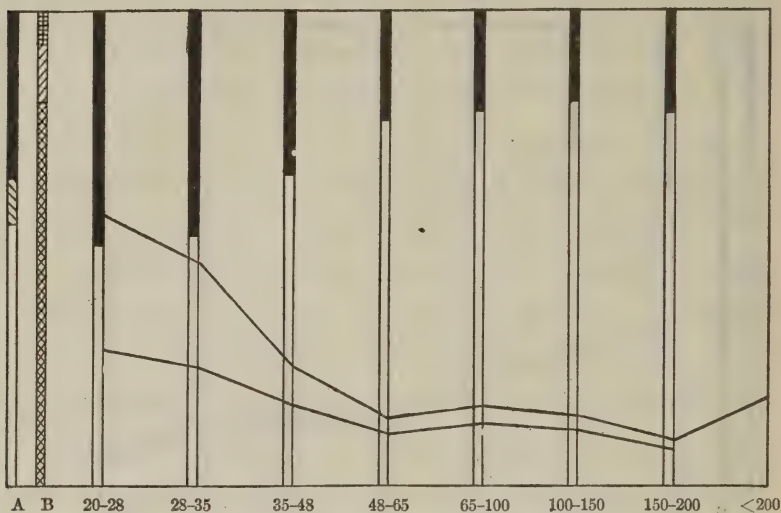


FIGURE 11.—Results of tests of sample 9.

No analyses were made of the products of this separation; but, as the ore was much finer grained than that from the overlying part

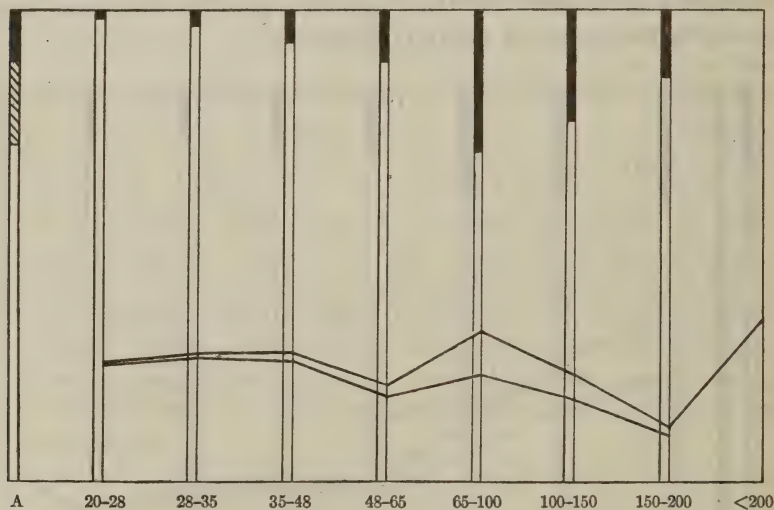


FIGURE 12.—Results of tests of sample 10.

of the bed, another portion of the sample was crushed to pass 35 mesh, and subjected to separation tests. The results are given on page 22.



As no analyses were made of the 20-mesh separation products, the composition of the products can not be compared. In the 35-mesh test the percentage of concentrates was reduced 8 per cent and percentage of tailings increased 6.4 per cent. Probably the concentrate was a little higher in grade and the slimes, on account of silica being crushed, a little lower. The screen-analysis curves show the same characteristics.

#### COMMENTS ON TESTS OF ORE FROM FRANK WHITE MINE.

In the separation tests the upper 6 feet of soft ore, averaging, in the two samples, 44.6 per cent Fe and 25.3 per cent  $\text{SiO}_2$ , gave a concentrate averaging 53 per cent Fe and 12.9 per cent  $\text{SiO}_2$ , with

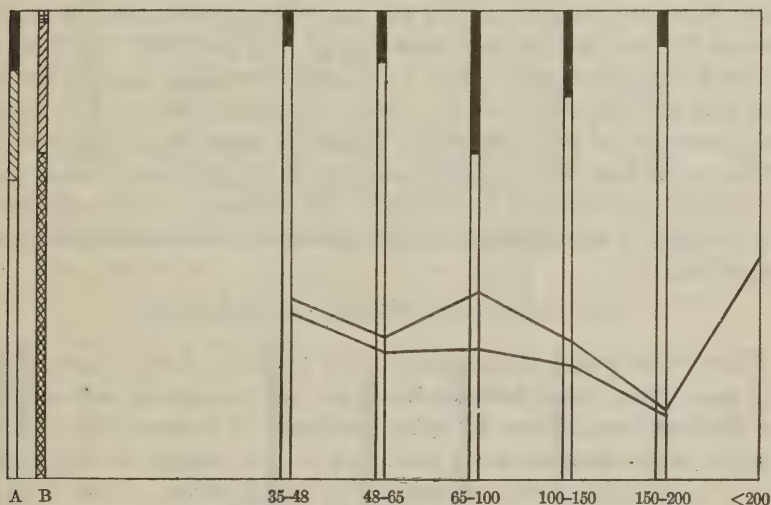


FIGURE 13.—Results of tests of sample 10 under 35 mesh.

a ratio of concentration of 1.42; adding the slimes to the concentrates gives a product averaging 53 per cent Fe and 12.9 per cent  $\text{SiO}_2$ , with a concentration ratio of 1.25 and an increase in iron recovery from 84.5 to 96.2 per cent. Adding the slimes to the concentrates materially increases the iron recovery without lowering the grade of the concentrate. The iron content of the crude ore is about 6 per cent below the average for the soft ore of the district and about 7 per cent higher than the average in silica, and the ore can be utilized only when the iron trade is active. The tests produced from it a product equaling in grade the best ores obtainable in the district.

The middle 6 feet of the Big seam is too low grade to be workable. The concentrates are poorer than the average soft ore of the district, and hence concentration by the method described would not be feasible for this part of the bed.

The composition of the lowest 6 feet of the bed is almost identical with that of the upper 6 feet, but the concentrates were no better than those from the middle 6 feet. The slimes from the lowest 6 feet are of much higher grade, however, and adding them to the concentrate gives a product slightly better than the average soft ore. The results are much poorer than those obtained with the upper 6 feet, and indicate that concentration of such ore is not practicable with the method used. The much finer grain of this ore, which prevents the silica from being separated to the same degree, accounts for the difference, although the ore was crushed to pass 35 mesh as against 20 mesh for ore from the upper 6 feet.

Averaging the analyses gives 42.2 per cent Fe and 30.5 per cent  $\text{SiO}_2$  for the total thickness of the seam. Concentrates from it would carry 50.9 per cent Fe and 17.2 per cent  $\text{SiO}_2$ , with a ratio of concentration of 1.60 and an iron recovery of 78.2 per cent. Adding the slimes to the concentrate gives a product averaging 51.1 per cent Fe and 16.8 per cent  $\text{SiO}_2$ , with a ratio of concentration of 1.31 and an iron recovery of 95.5 per cent. Hence it seems that concentration of the entire bed, with an iron recovery of 95.5 per cent, would yield a product considerably better than the average of the soft ore of the district. Concentration would, therefore, make the entire seam workable.

#### ORE FROM THE LOWER BENCH, BIG SEAM.

There were tested 23 samples of ore from the lower bench of the Big seam from mines between Bessemer and Graces Gap and one from the Ruffner mine, about 10 miles northeast of Graces Gap. The 23 samples were obtained from that part of the district in which mining is most active, and, consequently, beneficiation of the siliceous ores is of immediate importance. Hence these ores have received more attention in this investigation than other ores and other parts of the district.

#### SAMPLE 11. RUFFNER MINE.

Sample 11 represents the upper 8 feet of the lower unworkable part (11 feet thick) and was taken in 2 left heading, 30 feet from the slope. It has the same fine grain as ore from the upper part of the seam, this feature being clearly shown by the screen-analysis curves (figs. 14 and 15).

Because of the ore being fine grained, the separation products were not analyzed, but another part of the sample was crushed to pass 35 mesh and tested, with the results given on page 22.

As the 20-mesh products were not analyzed, their composition can not be compared. The 35-mesh size yielded 8.5 per cent less

concentrates, 5.6 per cent more slimes, and 2.9 per cent more tailings. This doubtless means that the concentrates of the 35-mesh size are

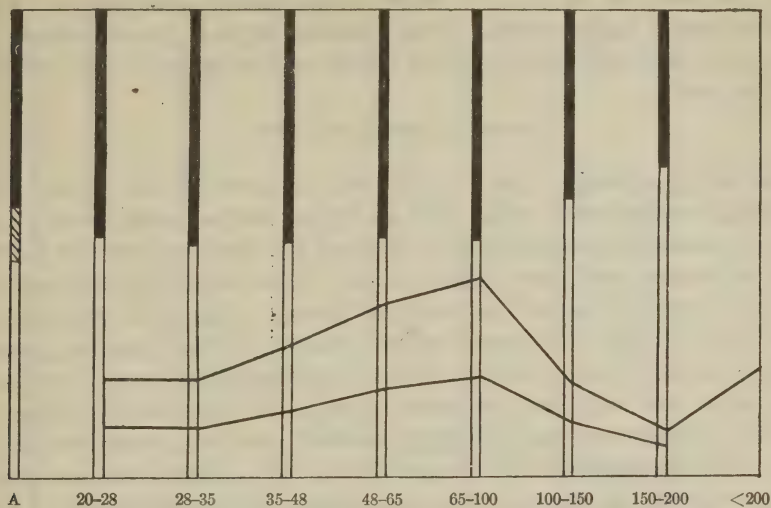


FIGURE 14.—Results of tests of sample 11.

appreciably better but the slimes are poorer than those of the 20-mesh size. The crude ore is of slightly lower grade than the two tested samples of crude ore from the upper part of the seam, yet the con-

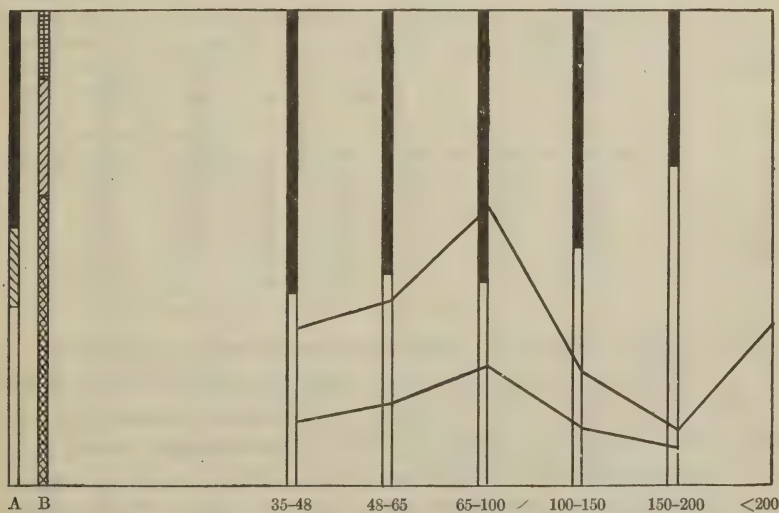


FIGURE 15.—Results of tests of sample 11 under 35 mesh.

centrate obtained runs 2 per cent higher in iron and nearly 1 per cent lower in silica than the average of that obtained from the other two samples; and the composition of concentrates plus slimes runs 2.2



per cent higher in iron and 1.3 per cent lower in silica than for the upper ores. As in the tests of the upper ore, however, the grade of the products is too low to make concentration attractive. The results would indicate that, if the Ruffner ore is to be concentrated, the entire bed will yield as good results as the upper 7 feet worked in the past.

SAMPLE 12. SPAULDING MINE.

In the Spaulding mine the Big seam is about 22 feet thick. No parting separates the seam into an upper and a lower bench, the upper more calcareous and less siliceous ore grading into the lower less calcareous and more siliceous ore. When the mine was active only the upper 11 feet of the seam was mined. Sample 12 represented the lower 11 feet of the seam and was taken in 1 left heading, cross left (a cross left is a branch to the left from a left main heading, and hence cuts well into the lower bench), 700 feet from the slope.

The concentrates and tailings from each mesh size were analyzed, and from the analyses the composition of the concentrates and tailings, as given on page 23, was calculated. The composition of the crude ore of each mesh size and the percentages of iron that went into the concentrates and tailings were calculated. The results are shown in the table below:

*Composition of concentrates and tailings by mesh sizes.*

Size of mesh.	Concentrates.		Tailings.		Crude.		Total iron in—	
	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Concentrates.	Tailings.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
20 to 28.....	42.2	15.6	8.6	75.5	29.4	33.5	88.8	11.2
28 to 35.....	44.1	12.7	8.6	73.1	32.2	33.0	91.0	9.0
35 to 48.....	45.0	10.8	8.9	64.5	36.1	24.0	93.9	6.1
48 to 65.....	47.0	9.2	9.3	54.2	38.5	19.3	94.6	5.4
65 to 100.....	48.6	8.9	8.5	54.0	39.9	18.7	95.4	4.6
100 to 150.....	53.1	7.5	8.9	48.7	43.5	16.4	95.6	4.4
150 to 200.....	52.4	9.6	14.9	42.4	45.1	16.0	93.5	6.5
200.....					46.1	14.0		

The 20 to 28 and 28 to 35 mesh sizes are of lower grade than the original crude ore, whereas the finer sizes are all of higher grade, the quality improving with the fineness. If the 20 to 28 mesh size were rejected, the concentrate obtained from the remainder would average 46.7 per cent Fe and 10.6 per cent SiO<sub>2</sub>, with a ratio of concentration of 2.07 and an iron recovery of 59.9 per cent. This product would run 1.2 per cent higher in iron and 1.3 per cent lower in silica, but the decrease in the iron recovery would be 23.2 per cent. Hence rejection of that size would not be warranted.



## SAMPLE 13. SPAULDING MINE.

Sample 13 represents the upper 6 feet of the lower bench in 3 left heading, at the breast.

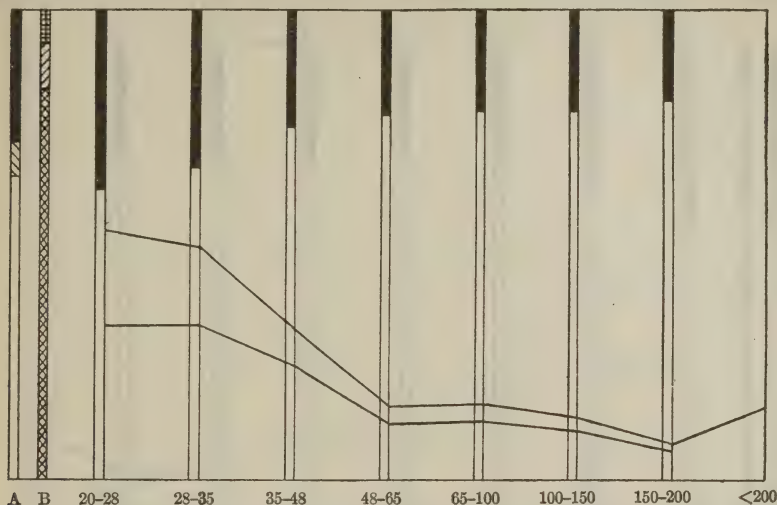


FIGURE 16.—Results of tests of sample 12.

## SAMPLE 14. SPAULDING MINE.

Sample 14 was taken from the upper 4 feet of the lower bench of the Big seam in 5 left heading, 1,600 feet from the slope.

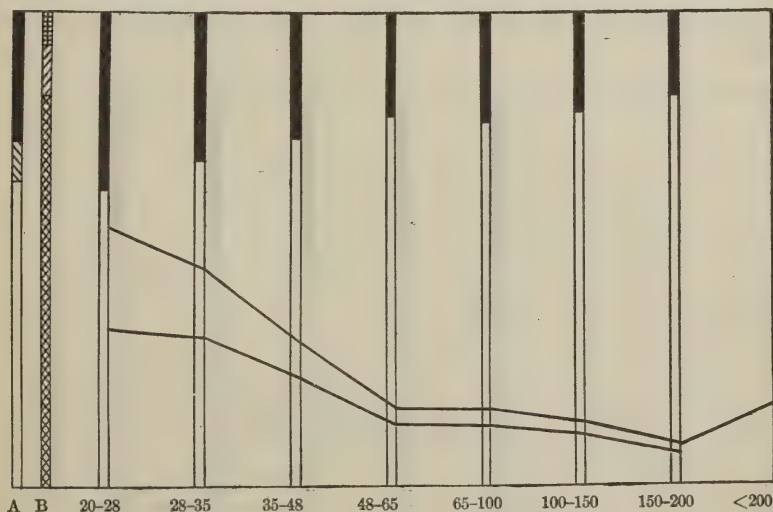


FIGURE 17.—Results of tests of sample 13.

Another part of the sample was crushed to pass 35 mesh and subjected to separation tests, with the results given on page 24. Only

the concentrates were analyzed, they contained 45.3 per cent Fe and 13.0 per cent  $\text{SiO}_2$ , the ratio of concentration being 1.90 and the iron recovery 67.7 per cent. Compared with the 20-mesh test these concentrates have only 0.2 per cent more iron, and only 0.5 per cent less

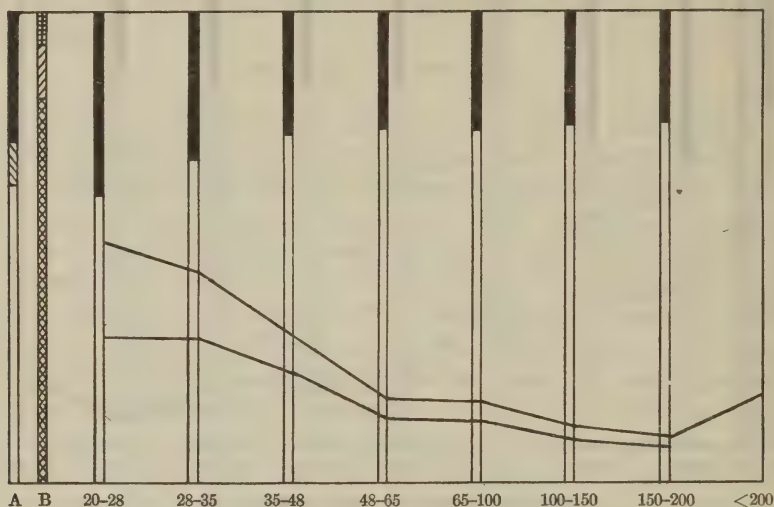


FIGURE 18.—Results of tests of sample 14.

silica, although the iron recovery is 13.7 per cent less. Such results would not justify the finer grinding. The amount of tailing is nearly the same for both 20 and 35 mesh ore, so that the lower percentage of concentrates is due almost entirely to the higher percentage of slimes.

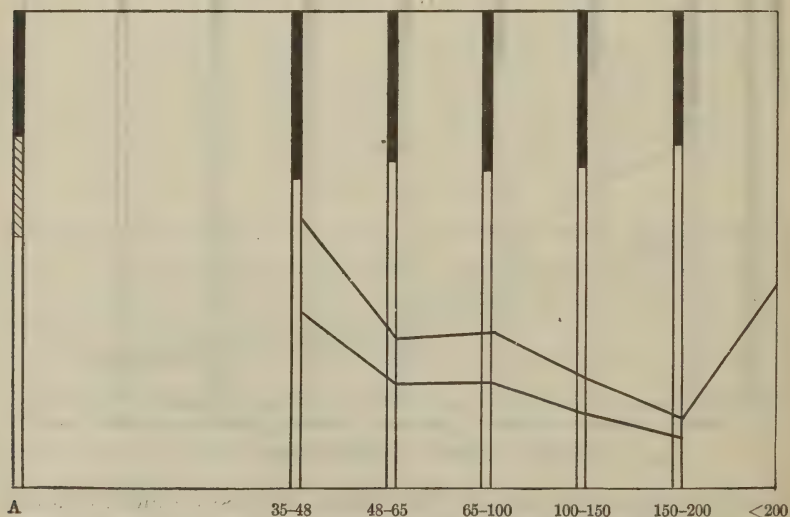


FIGURE 19.—Results of tests of sample 14 under 35 mesh.

## SAMPLE 15. SPAULDING MINE.

Sample 15 was taken from the upper 3 feet of the lower bench of the Big seam in 6 left heading, at the breast of the drift.

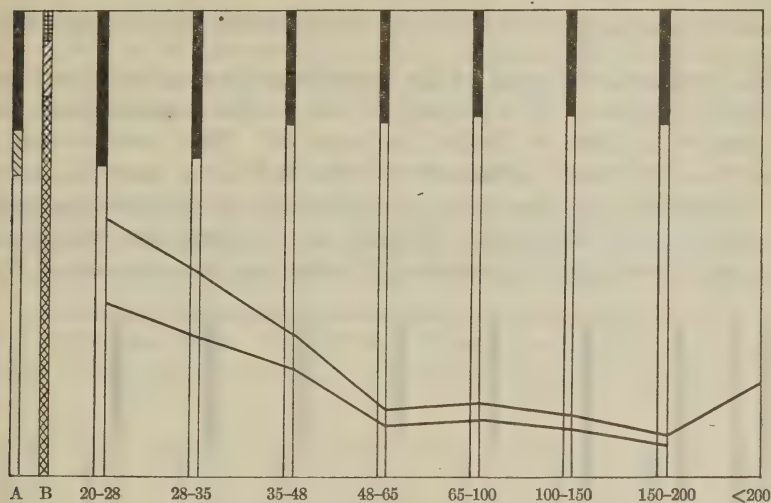


FIGURE 20.—Results of tests of sample 15.

## SAMPLE 16. SPAULDING MINE.

Sample 16 was taken from the 3 feet of ore immediately below the workable ore, in 10 left heading, at the breast.

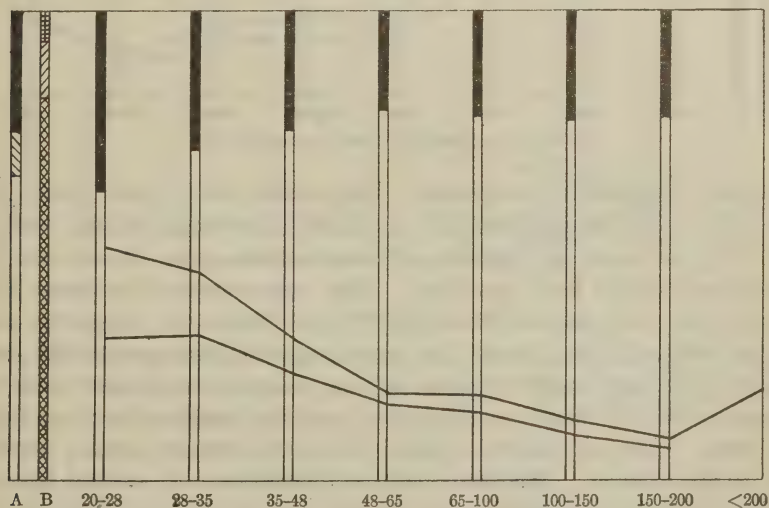


FIGURE 21.—Results of tests of sample 16.

## SAMPLE 17. SPAULDING MINE.

Sample 17 was taken from the upper 8 feet of the lower bench of the Big seam in 6 right heading, 600 feet from the slope.

## COMMENTS ON TESTS OF LOWER BENCH BIG SEAM ORE FROM THE SPAULDING MINE.

The results of the tests of the preceding six samples of ore are strikingly uniform. The average of the results obtained from the six samples is given in Table 1 on page 38. They deviate only a little from the results obtained with the individual samples, hence the Spaulding lower bench ore is presumably of uniform character throughout, and the results obtained may be regarded with more than usual confidence as representing what may be expected of this

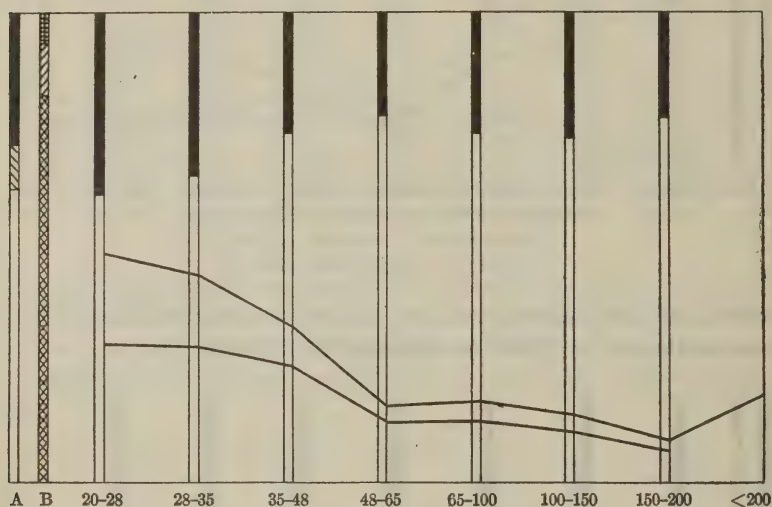


FIGURE 22.—Results of tests of sample 17.

ore under large-scale treatment. The slimes are so little lower in grade than the concentrates that in order to increase the iron recovery 11.4 per cent the mixture of concentrates plus slimes can be considered as the final product. Then concentration increases the iron 9.5 per cent, reduces the silica 13.7 per cent, and, judged by the single determination, leaves the percentage of lime practically unchanged, the net result being a decided increase in quality, with a concentration of about 4 tons into 3 and an iron recovery of 93.3 per cent. Such a product is better than the average washed brown ore of the district. Burchard<sup>a</sup> gives 23 analyses of washed brown

<sup>a</sup> Burchard, E. F., and Butts, Charles, Iron ores, fuels, and fluxes of the Birmingham district, Ala.: U. S. Geol. Survey Bull. 400, 1910, p. 169.



ore from the Woodstock and Champion areas, which average 45 per cent Fe, 14.6 per cent  $\text{SiO}_2$ , 4 per cent  $\text{Al}_2\text{O}_3$ , and 0.74 per cent Mn. The Spaulding concentrate runs 0.6 per cent higher in iron and 1.5 per cent lower in silica and its lime content more than counterbalances the manganese content of the brown ore. According to the results, concentration of the lower bench Spaulding ore should produce a product that can easily compete with the brown ores of the district.

#### SAMPLE 18. ISHKOODA MINE.

The upper bench in the Ishkooda mine is about 7 feet thick. A mere parting along the bedding plane with only here and there a thin layer of shale separates it from the lower bench. In working this bench about 3 feet and sometimes as much as 5 feet of the lower bench ore is also taken. From 7 to 10 feet of ore is left beneath and sample 18 was taken from it in 18 left heading, 200 feet from the slope.

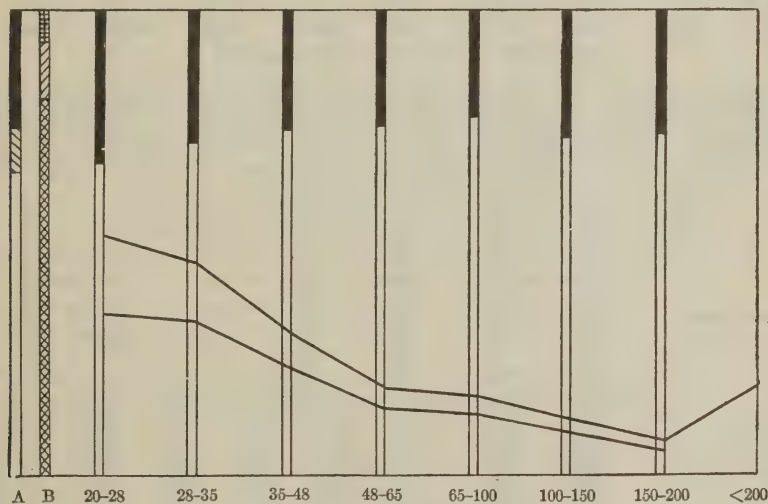


FIGURE 23.—Results of tests of sample 18.

#### SAMPLE 19. ISHKOODA MINE.

Sample 19 was taken from the upper 4 feet of the lower bench of the Big seam at the same place as sample 18 and represents lower bench ore that is mined with upper bench ore.

#### SAMPLE 20. ISHKOODA MINE.

Sample 20 was taken in 15 right heading, 500 feet from the slope, and represents the lower bench 8 to 10 feet below the parting.

The concentrates and tailings of each mesh size were analyzed separately and the composition of the concentrates and tailings for

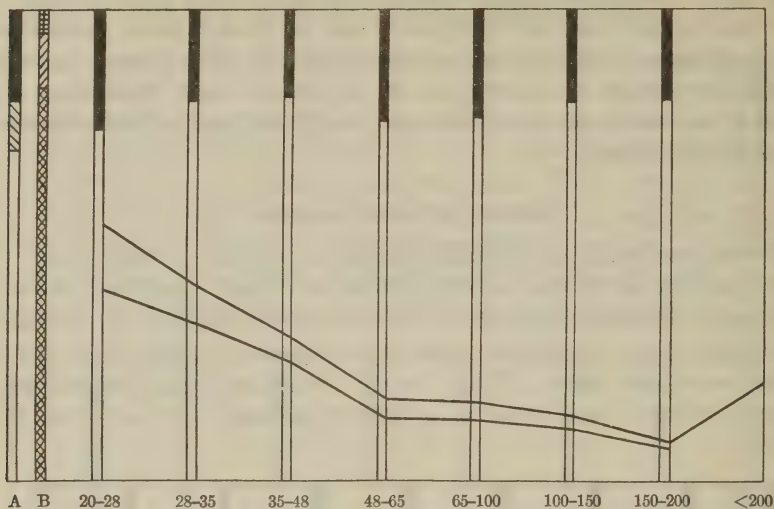


FIGURE 24.—Results of tests of sample 19.

the entire sample, as given in Table 1 on page 26, was calculated from these. The table below shows these analyses and the calculated composition of the crude of each mesh size.

*Composition of concentrates and tailing by mesh sizes.*

Size of mesh.	Concentrates.		Tailings.		Crude.		Total iron in—	
	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Concen- trates.	Tail- ings.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
20 to 28.....	43.5	19.3	8.7	68.5	28.8	40.1	87.3	12.7
28 to 35.....	37.2	21.7	9.1	63.3	29.6	33.1	91.6	8.4
35 to 48.....	42.2	12.4	8.7	51.8	31.8	24.7	91.5	8.5
48 to 65.....	43.3	10.4	8.1	41.6	32.4	20.1	92.3	7.7
65 to 100.....	46.6	9.2	8.0	36.8	33.9	18.3	92.2	7.8
100 to 150.....	47.9	9.7	8.9	33.3	36.3	16.7	92.7	7.3
150 to 200.....	50.7	8.6	13.1	29.3	43.3	12.7	94.0	6.0
200.....					39.8	12.4		

The 20 to 28 and 28 to 35 mesh sizes are of lower grade than the original ore, whereas all smaller sizes of the crude are of higher grade, the quality increasing with the fineness, except that the 150 to 200 mesh size contains more iron than the slimes, and is higher in iron and lower in silica than the concentrates themselves. Hence, everything smaller than 150-mesh might have been added directly to the final product.

Another part of the same sample was crushed to pass 35-mesh and subjected to separation tests, with the results given on page 26. By

analysis these concentrates contained 42.1 per cent Fe and 13.3 per cent  $\text{SiO}_2$ , the silica being nearly 3 per cent lower than in the con-

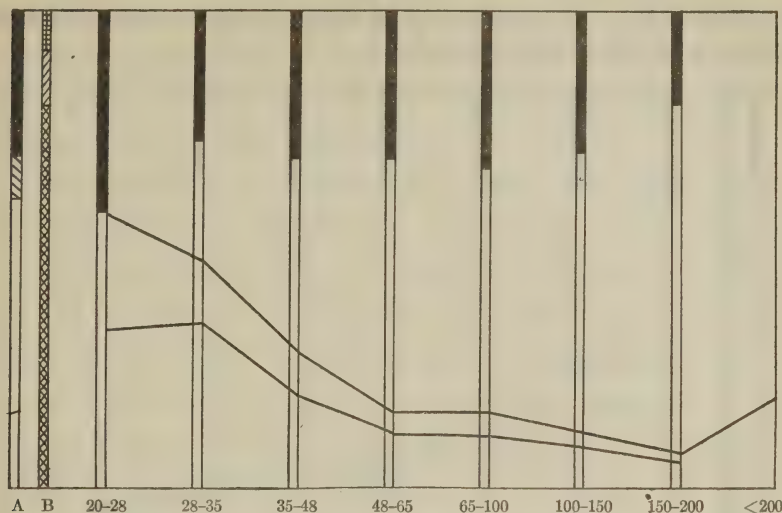


FIGURE 25.—Results of tests of sample 20.

centrates of the 20-mesh test. Compared with that test the ratio of concentration is 2.10 instead of 1.66, and the iron recovery 61.9 as against 80 per cent. Finer crushing increased the proportion of

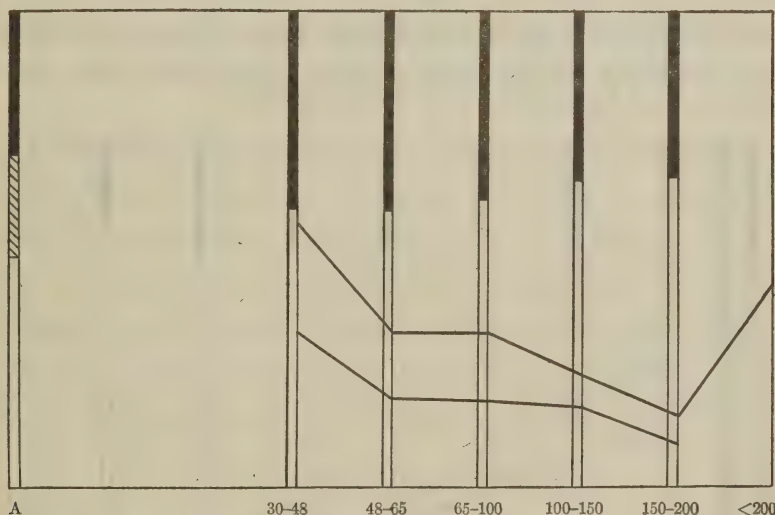


FIGURE 26.—Results of tests of sample 20 under 35 mesh.

tailings only a little more than 1 per cent, and the decrease of concentrates, 12.8 per cent, is offset by the larger proportion of slimes. Adding the slimes to the concentrates yields a final product that would probably differ little from the corresponding product of the 20-mesh test.

## SAMPLE 21. ISHKOODA MINE.

Sample 21 was taken in 6 right heading, near the manway. Here the parting is a 1 to 2 inch layer of shale, and the sample was taken from the  $3\frac{1}{2}$  feet of ore just under it.

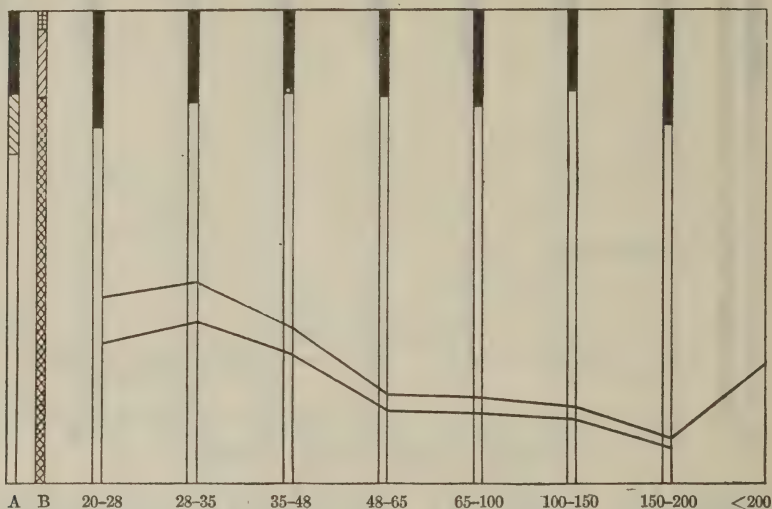


FIGURE 27.—Results of tests of sample 21.

## SAMPLE 22. ISHKOODA MINE.

Sample 22 was taken from the upper 7 feet of the lower bench in 10 right heading, 200 feet from the slope. Here the parting is merely a break in the bedding, no shale being present.

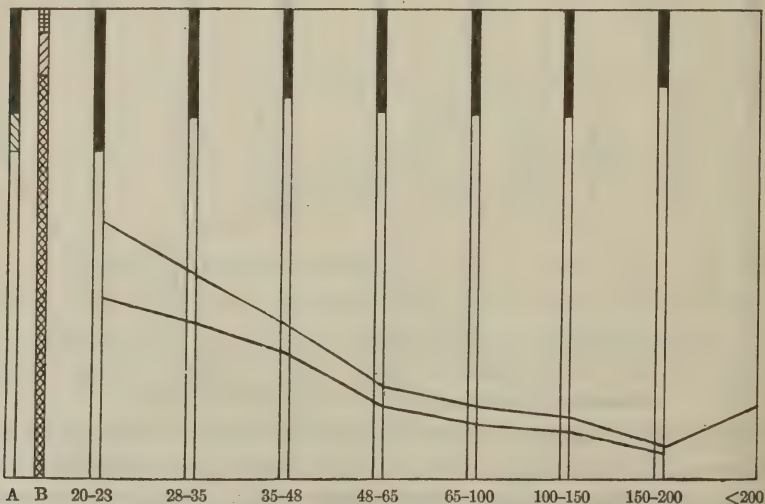


FIGURE 28.—Results of tests of sample 22.



## COMMENTS ON TESTS OF ORE FROM LOWER BENCH IN ISHKOODA MINE.

Compared with the Spaulding ore, the five samples show considerable variation, largely because samples 20 and 21 represent only the less siliceous part of the lower bench ore that is taken out with the upper bench in mining. In consequence the averages of the five samples of Ishkooda ore, on page 37, are somewhat better than the probable average of the whole bench.

The composition of the slimes is so nearly like that of the concentrates that only a final product of slimes plus concentrates need be considered, this product being of the same grade and the iron recovery being higher. This product is 8.5 per cent higher in iron and 13.3 per cent lower in silica than the original ore and has about the same lime content according to the one determination. Compared with the average of the brown ores of the district (p. 76), this product runs 0.6 per cent higher in iron, 2.3 per cent lower in silica, and has a lime content of nearly 8 per cent to offset the manganese content of the brown ores. Judged by its composition, it can readily compete with them. As samples 19 and 21 represent only the best part of the lower bench, the results from samples 18, 20, and 22 were also averaged separately. The composition of the concentrates plus slimes averaged 44.0 per cent Fe and 13.6 per cent  $\text{SiO}_2$ , or 1 per cent less in iron and silica than the brown ores, and hence compares favorably with them. This result is obtained with a concentration of 4 tons into 3 and an iron recovery of 93.2 per cent.

## SAMPLES FROM SONGO MINE.

In this mine the upper bench of the Big seam is  $7\frac{1}{2}$  to 9 feet thick and is separated from the lower bench by a 6 to 9 inch shale parting. The lower bench is about 12 feet thick, its upper 9 feet being relatively free from shale. At only one point, in 14 right heading, was the lowest 3 feet cut into, and there it consisted of intercalated shale and ore. Down to heading 23, the upper 9 feet of the lower bench has been cut into in the headings and the ore used by the Woodward Iron Co., which works this mine, in its own furnaces; but the quality is such that the ore can not be sold in the open market. Nearly all the ore mined is from above the shale parting.

## SAMPLE 23. SONGO MINE.

Sample 23 was taken in 14 right heading, about 600 feet from the slope; at this point the entire lower bench is exposed, but on account of the lower 3 feet being shaly only the upper 9 feet were sampled.

## SAMPLE 24. SONGO MINE.

Sample 24 represents the upper 6 feet of the lower bench in 12 left heading, 250 feet from the slope.

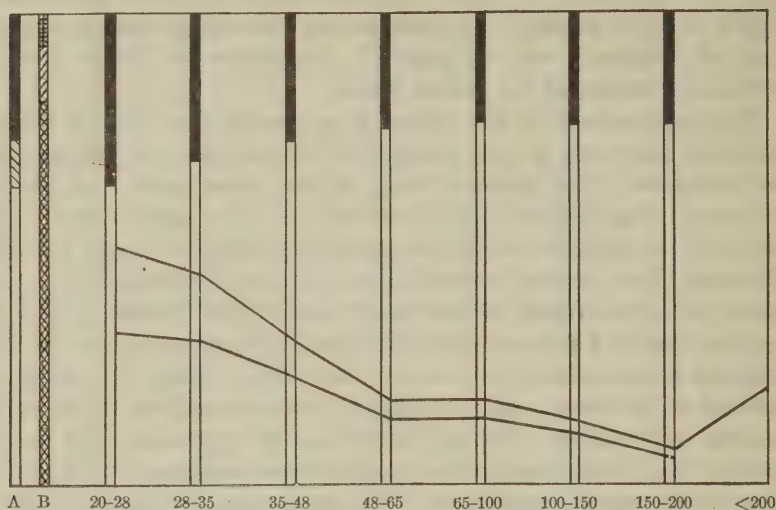


FIGURE 29.—Results of tests of sample 23.

## SAMPLE 25. SONGO MINE.

Sample 25 was taken from the upper 9 feet of the lower bench in 15 left heading, 300 feet from the slope.

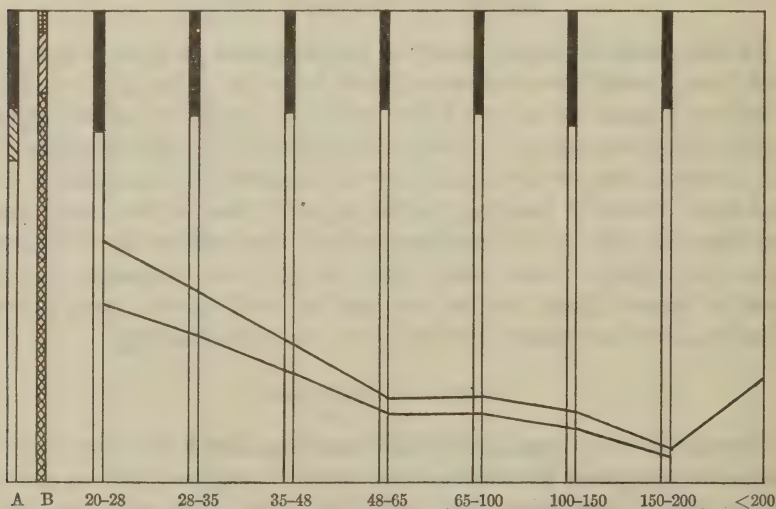


FIGURE 30.—Results of tests of sample 24.

## SAMPLE 26, SONGO MINE.

Sample 26 was taken from the upper 9 feet of the lower bench of the Big seam in 18 right heading, 700 feet from the slope.

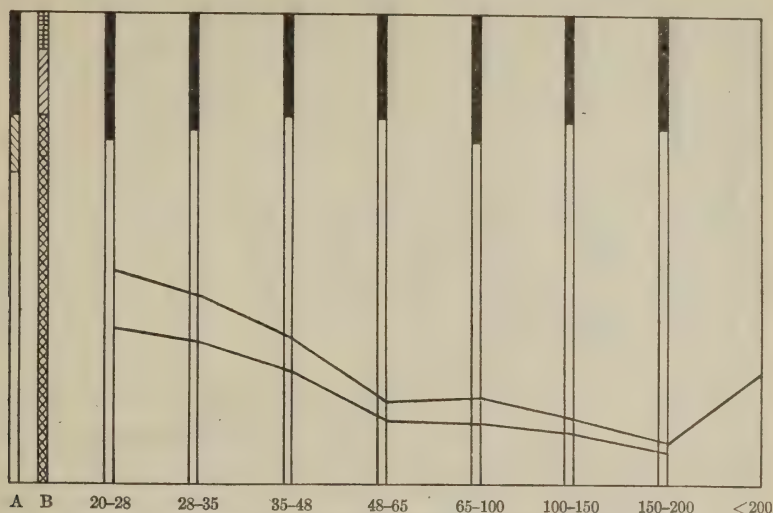


FIGURE 31.—Results of tests of sample 25.

## SAMPLE 27, SONGO MINE.

Sample 27 was taken from the upper 9 feet of the lower bench in 19 left heading, 300 feet from the slope.

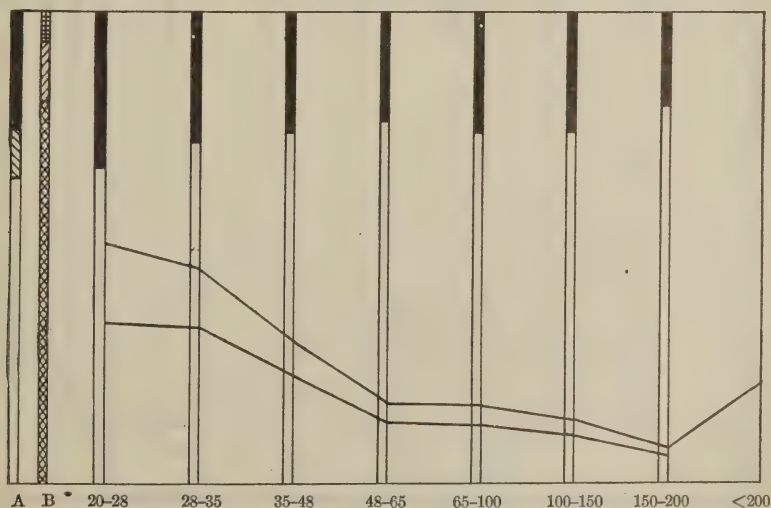


FIGURE 32.—Results of tests of sample 26.

Another portion of the sample was crushed to pass 35 mesh and subjected to separation tests, with the results given on page 29

The concentrates from this separation analyzed 45.7 per cent Fe and 11.0 per cent  $\text{SiO}_2$ , with a ratio of concentration of 1.79 and an iron recovery of 68.5 per cent. Finer crushing consequently produced a

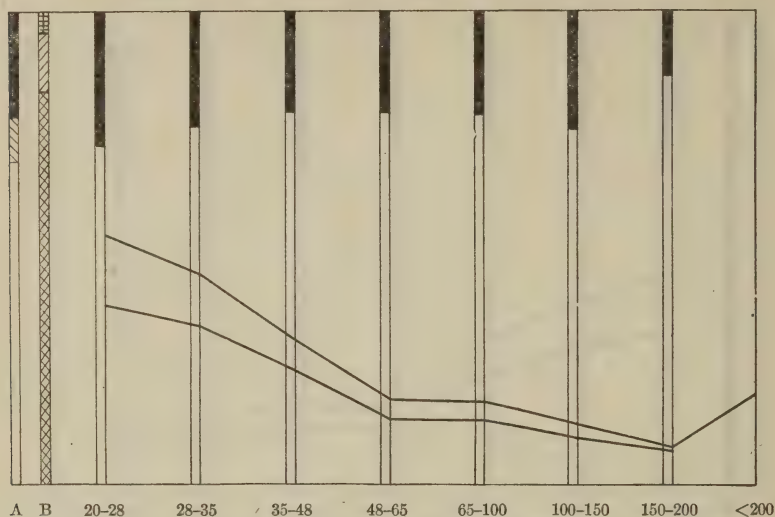


FIGURE 33.—Results of tests of sample 27.

concentrate 2.2 per cent higher in iron and lower in silica, but reduced the iron recovery from 83.2 to 68.5 per cent. The proportion of tailings was nearly the same in the two tests, and the slimes increased at the expense of the concentrates.

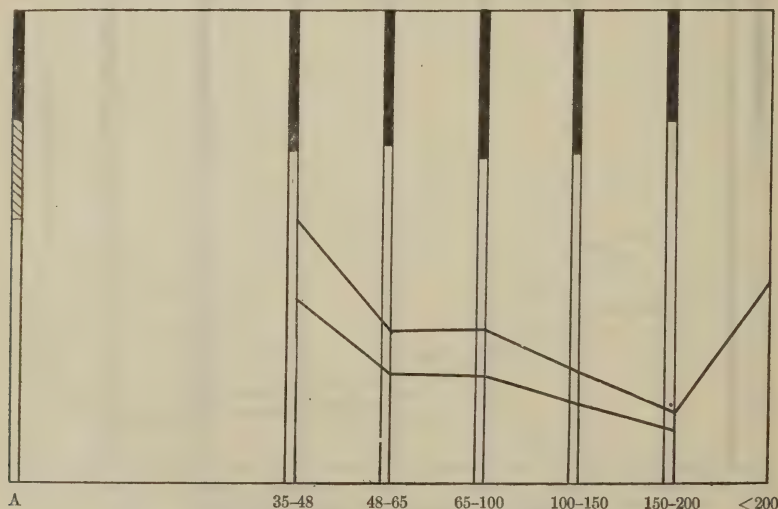


FIGURE 34.—Results of tests of sample 27 under 35 mesh.

SAMPLE 28. SONGO MINE.

Sample 28 was taken from the upper 8 feet of the lower bench of the Big seam in 23 left heading, 150 feet from the slope.



The concentrates and tailings of each mesh size were analyzed separately and the composition of the entire concentrates and tailings calculated from these analyses, which are given in the table below, with the calculated composition of the crude ore of each size.

*Composition of concentrates and tailings by mesh sizes.*

Size of mesh.	Concentrates.		Tailings.		Crude.		Total iron in—	
	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Concentrates.	Tailings.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
20 to 28.....	41.5	15.4	8.3	65.3	31.9	29.9	92.4	7.6
28 to 35.....	43.8	12.4	8.2	62.6	35.2	24.4	94.4	5.6
35 to 48.....	44.6	11.1	8.1	54.7	37.3	19.9	95.6	4.4
48 to 65.....	54.8	9.9	8.3	46.8	38.3	17.3	95.7	4.3
65 to 100.....	49.5	8.8	7.9	43.1	38.7	16.2	95.6	4.4
100 to 150.....	53.0	7.7	6.9	42.9	43.0	15.2	96.6	3.4
150 to 200.....	54.4	8.1	7.1	41.8	45.3	14.6	97.0	3.0
200.....					47.4	11.7		

Mesh sizes larger than 48 mesh are of lower grade and those smaller than 48 mesh are of higher grade than the original ore, each size being of higher grade than the preceding. Of the crude ore sizes only the slimes are better than the concentrates, but the crude 150 to 200 mesh ore runs better than the concentrate of the 20 to 28 mesh size.

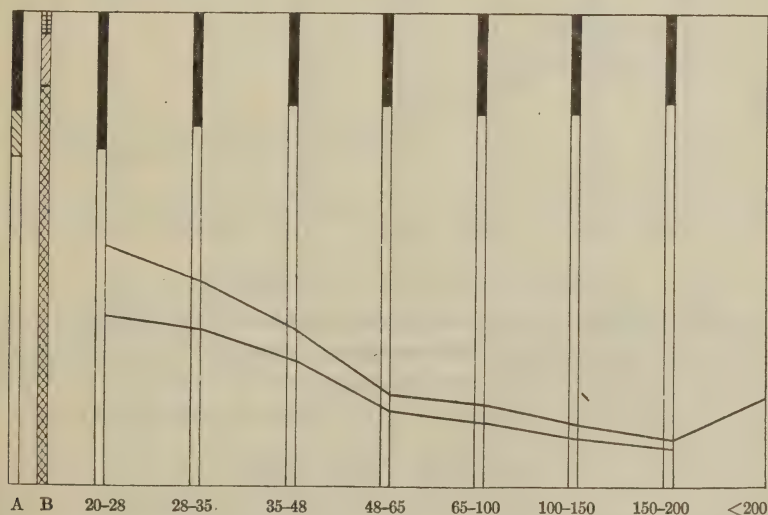


FIGURE 35.—Results of tests of sample 28.

#### COMMENTS OF TESTS OF ORE FROM SONGO MINE.

The six samples from the lower bench show a uniformity of composition almost equal to that of the Spaulding ore. Hence the bed at the points of sampling is of uniform grade chemically, and also, as

shown by the separation tests, physically. Such uniformity of character indicates that in actual practice there would probably be little deviation from these results. The average of the results obtained with the six samples is given in the table on page 38.

An average of several analyses supplied by Mr. A. H. Woodward of lower bench Songo ore is 39.5 per cent Fe, 20 per cent insoluble, and 10.4 per cent CaO. These figures indicate that the samples tested are representative of the ore. The slimes practically average the same as the concentrates, therefore, in view of the increased iron recovery, the final product may be considered as concentrates plus slimes. Such a product on the basis of 4 tons into 3, will give an iron recovery of 94.3 per cent, and be 6.6 per cent higher in iron, 8.7 per cent lower in silica, and nearly as high in lime as the crude ore.

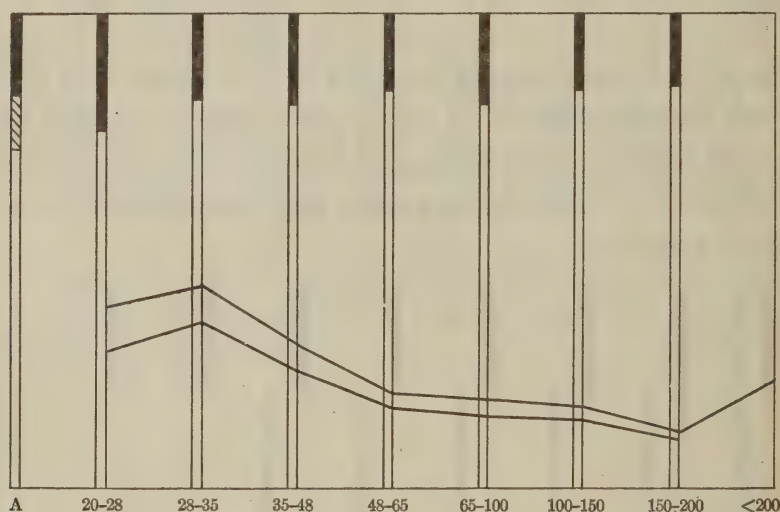


FIGURE 36.—Results of tests of sample 29.

Compared with the average of the brown ores of the district (p. 76), the product has the same iron content, 2 per cent less silica, and a high lime content to offset the manganese of the brown ores. On the basis of its chemical composition it can compete with those ores.

#### ORE FROM FOSSIL MINE.

#### SAMPLE 29. FOSSIL MINE.

In this mine the upper bench only of the Big seam is worked. Sample 29 represents the upper 4 feet of the lower bench, in 10 slope, 24 left heading, 100 feet from the slope.

Only the concentrates were analyzed, and they carried 48 per cent Fe and 12.1 per cent  $\text{SiO}_2$ . Another portion of the same sample was crushed to pass 35 mesh and subjected to separation tests, with the results given on page 30.

As the composition of the slimes is so much lower than that of the concentrates, adding the slimes to the concentrates would lower the grade of the final product. Yet as the iron recovery would be increased and the resulting product would still be slightly higher in iron and lower in silica than the average of the brown ores, the procedure would probably be found practicable. The 20 mesh concentrate is practically as good as the 35 mesh, and was obtained with a ratio of concentration of 1.40 as against 1.71, and an iron recovery of 86.2 per cent as against 71.1 per cent. There seems to be no advantage, therefore, in crushing this ore finer than 20 mesh. At least, 35 mesh is not fine enough to show an appreciable advantage.

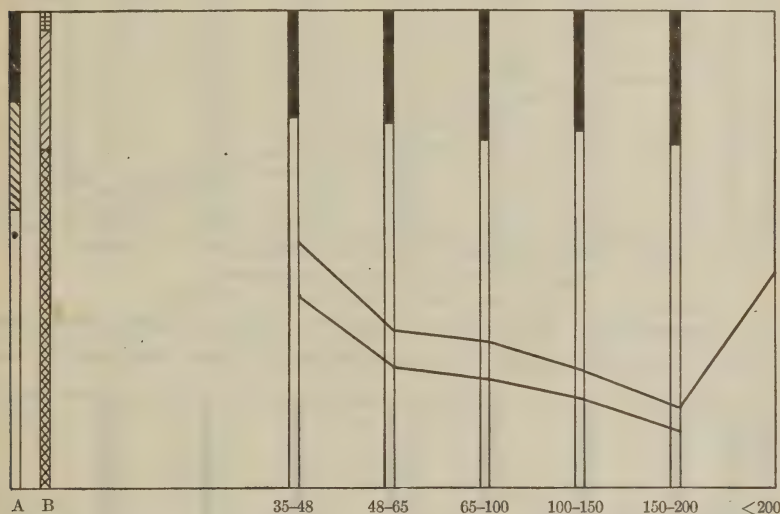


FIGURE 37.—Results of tests of sample 29 under 35 mesh.

#### SAMPLE 30. FOSSIL MINE.

Sample 30 was taken in 10 slope, 31 left heading, 125 feet from the slope. The lower bench here is a little more than 8 feet thick, and the sample represents the upper 8 feet of it.

#### SAMPLE 31. FOSSIL MINE.

Sample 31 was taken at 31 heading in the bottom of 10 slope. At that place the parting of the two benches comprises two  $\frac{1}{2}$ -inch bands of shale, with a 4-inch layer of ore between. The lower bench is 9 feet thick, and the sample represents the entire thickness.

#### SAMPLE 32. FOSSIL MINE.

Sample 32 represents the entire thickness of the lower bench in  $9\frac{1}{2}$  slope, 23 left heading, 30 feet from the slope.

## SAMPLE 33. FOSSIL MINE.

Sample 33 represents the whole thickness of the lower bench, on the north side of 9 slope between headings 47 and 49.

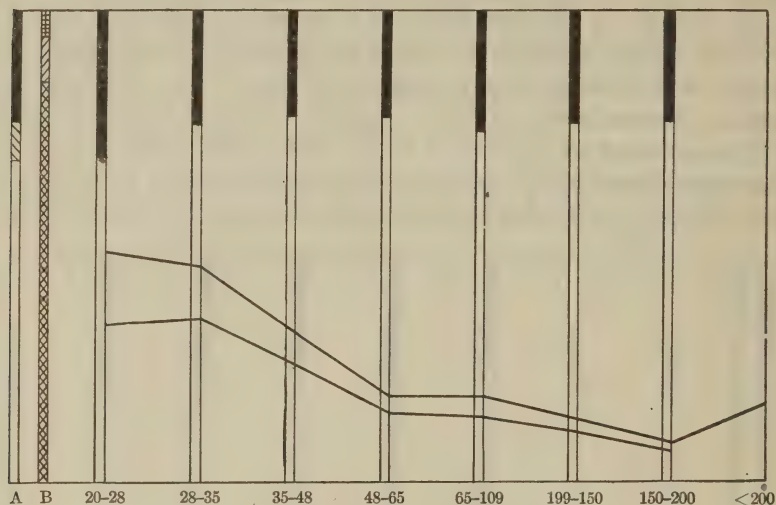


FIGURE 38.—Results of tests of sample 30.

The concentrates and tailings from the different mesh sizes were analyzed separately, and the composition of the concentrates and

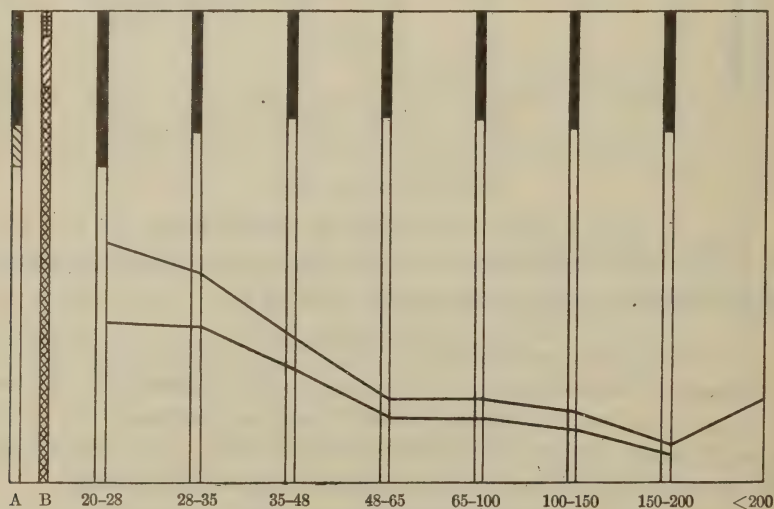


FIGURE 39.—Results of tests of sample 31.

tailings, as given on page 31, was calculated from these analyses. The calculated composition of the crude ore of each size and the per-



centage of iron that went into the concentrates and tailings of each are given in the accompanying table.

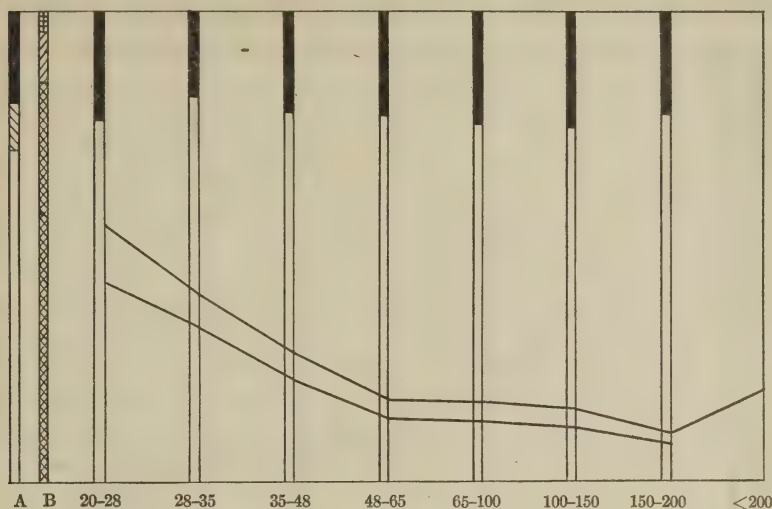


FIGURE 40.—Results of tests of sample 32.

*Composition of concentrates and tailings by mesh sizes.*

Size of mesh.	Concentrates.		Tailings.		Crude.		Total iron in—	
	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Concentrates.	Tailings.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
20 to 28.....	42.4	15.6	8.2	70.9	32.1	32.3	92.3	7.7
28 to 35.....	44.4	12.7	8.6	63.3	36.7	23.8	94.9	5.1
35 to 48.....	44.3	11.5	8.5	53.5	37.4	19.7	95.6	4.4
48 to 65.....	45.9	9.9	7.9	41.9	38.6	16.0	96.0	4.0
65 to 100.....	48.3	8.9	8.0	37.1	39.9	14.8	95.8	4.2
100 to 150.....	50.6	8.6	10.2	30.9	42.4	13.2	95.1	4.9
150 to 200.....	54.9	7.5	10.4	32.9	44.7	13.3	94.7	5.3
200.....					45.5	11.3		

In this sample the 35 to 48 mesh size has about the same grade as the crude ore, and the coarser sizes are poorer. The finer sizes improve in quality down to the fines. The slimes are the only size of the crude ore that runs better than the concentrates, but all the crude smaller than 100 mesh runs better than the concentrates of the 20 to 28 mesh size.

SAMPLE 34. FOSSIL MINE.

Sample 34 represents the entire lower bench in 8 slope between headings 48 and 50.

## COMMENTS ON TESTS OF ORE FROM FOSSIL MINE.

As the results for sample 29 were obtained by crushing the ore to pass 35 mesh, and those for the rest of the samples by crushing to

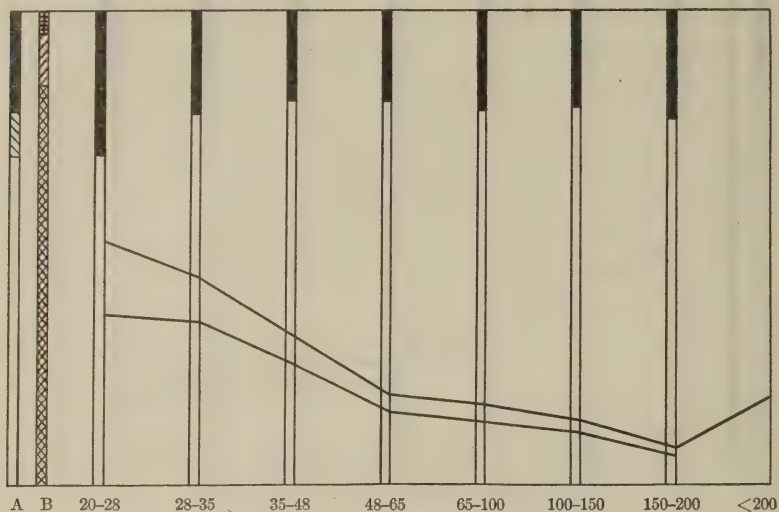


FIGURE 41.—Results of tests of sample 33.

pass 20 mesh, they are omitted in considering the average results (p. 39). The crude ore has the same composition as the other five

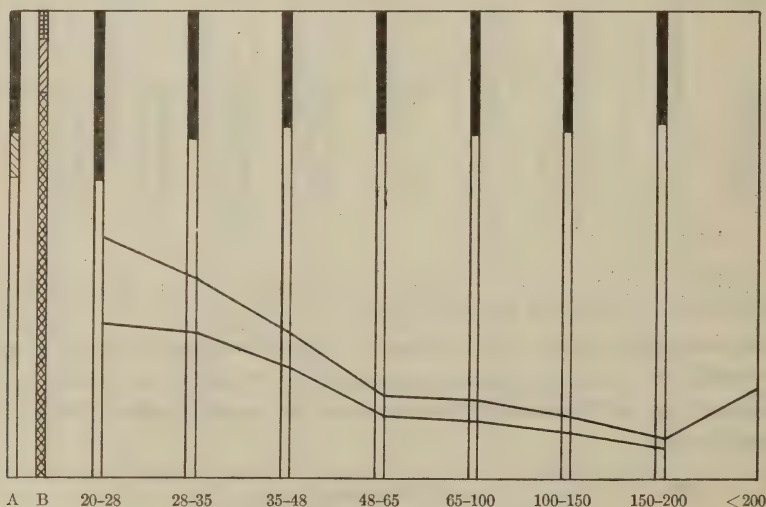


FIGURE 42.—Results of tests of sample 34.

samples, and the better products doubtless result from finer crushing, although the concentrate of the 20-mesh test runs a little higher than the concentrates from the rest of the samples.

An average of eight analyses of lower bench Fossil ore given by Burchard<sup>a</sup> is 37.3 per cent Fe, 20.2 per cent SiO<sub>2</sub>, 3.3 per cent Al<sub>2</sub>O<sub>3</sub>, and 11.9 per cent CaO. This is very nearly the same as the average of samples 30 to 34, as given in Table 2 on page 39. The individual samples depart little from the average and show the uniformity of results that characterizes the separation tests. The ore is, therefore, shown to be quite uniform in chemical composition and physical character throughout the mine.

As the grade of the slimes is so little below that of the concentrates, it is advantageous to consider the final product as concentrates plus slimes. Then a concentration in the ratio of 4 tons into 3 yields an iron recovery of 94.8 per cent and a product 7.7 per cent higher in iron, 8.6 per cent lower in silica, and, on the basis of the one analysis, only 2 per cent lower in lime. Compared with the average of the washed brown ores of the district, such a concentrate would be 0.3 per cent richer in iron, 2.4 per cent lower in silica, and have an appreciable lime content to offset the value of the manganese in the brown ore. Hence, on the basis of its chemical composition it should be able to compete favorably with those ores.

#### SUMMARY OF COMMENTS ON TESTS OF LOWER BENCH BIG SEAM ORE.

On pages 50-71 are given the results of tests with 24 samples of ore from the lower bench of the Big seam. Sample 11 from the Ruffner mine represents very low grade ore and its wet concentration under present conditions does not seem justifiable.

The remaining 23 samples represent ore from different points in four important mines. Sample 29, from the Fossil mine, was crushed to 35 mesh, and the complete results are for that mesh size; as complete results for other samples were determined with the 20-mesh crushing, sample 29 is omitted from the discussion of the average results. Finer crushing of sample 29 had little effect, however, in improving the grade of the concentrate, as the 20-mesh concentrate contained only 0.1 per cent less iron and 0.4 per cent more silica than ore crushed to pass 35 mesh, and the iron recovery was 15 per cent higher. Hence there was no advantage in crushing this ore finer.

The average of the results obtained with the remaining 22 samples is given in the accompanying table. The column marked concentrates included the concentrates plus slimes.

---

<sup>a</sup> Burchard, E. F., and Butts, Charles, Iron ores, fuels, and fluxes of the Birmingham district, Alabama, with chapters on the origin of the ores, by E. C. Eckel: U. S. Geol. Survey Bull. 400, 1910, pp. 83-84.

*Average results of tests.*

Mine.	Crude ore.			Concentrates.			Recovery of iron.	Ratio of concentration.
	Fe.	SiO <sub>2</sub> .	CaO.	Fe.	SiO <sub>2</sub> .	CaO.		
	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	
Spaulding <sup>a</sup> .....	36.1	26.8	7.5	45.6	13.1	7.6	93.3	1.37
Ishkooda <sup>b</sup> .....	37.1	25.6	8.0	45.6	12.3	7.7	94.1	1.31
Songo <sup>a</sup> .....	38.4	21.2	10.5	45.0	12.5	9.5	94.3	1.30
Fossil <sup>b</sup> .....	37.6	20.8	9.6	45.3	12.2	7.6	94.8	1.30
Average of the four groups.....	37.3	23.6	8.9	45.4	12.5	8.1	94.1	1.32
Average composition of washed brown ores.....				45.0	14.6			

<sup>a</sup> Average of six samples.    <sup>b</sup> Average of five samples.

As the results from the 22 samples are decidedly uniform, the calculated average in the table may be safely taken as representing what one may expect from beneficiating these ores under the conditions of the experiments. The final product obtained by concentrating 4 tons of crude ore into 3, with an iron recovery of 94.1 per cent, runs 0.4 per cent higher in iron, and 2.1 per cent lower in silica than the brown ores, and has a considerable lime content to enhance its value.

## ORE FROM THE IRONDALE SEAM.

The Irondale seam is of most value along the northeastern part of Red Mountain, from Pilot Knob southwest to a point about opposite Birmingham, a distance of about 10 miles. Farther southwest the bed becomes poorer, showing interstratified bands of low-grade ore and shale or losing its identity entirely. Burchard<sup>a</sup> says of the grade of the ore:

Its soft ore, now nearly all mined out either by surface trenches or drifts, is the best of the district. Its hard ore is also of high grade and has hitherto been for the most part held in reserve, since ore could be produced from the thicker Big seam at a lower cost per unit of iron.

The seam lies only a few feet below the Big seam and has a thickness of 4 to 6 feet.

Three samples from this seam were tested—two from the Ruffner mine and the third from the Hammond mine—they being from the middle part of the belt within which the seam has its maximum value.

## SAMPLE 35. RUFFNER MINE.

Sample 35 represents the entire thickness of the Irondale seam in 1 slope, 18 left heading, 650 feet from the slope. The bed here is 4 to 4½ feet thick.

<sup>a</sup> Burchard, E. F., and Butts, Charles, Iron ores, fuels, and fluxes of the Birmingham district, Alabama, with chapters on the origin of the ores, by E. C. Eckel: U. S. Geol. Survey Bull. 400, 1910, p. 47.



## SAMPLE 36. RUFFNER MINE.

Sample 36 represents the entire thickness of the Irondale seam in 1 slope, 22 right heading, 450 feet from the slope. The seam here is 4 feet thick.

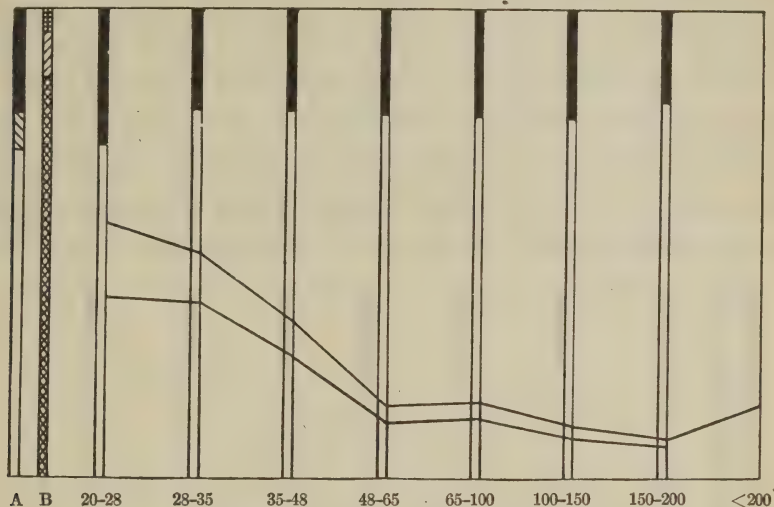


FIGURE 43.—Results of tests of sample 35.

## SAMPLE 37. HAMMOND MINE.

Sample 37 was taken in the slope on the Irondale seam at the point where it is cut by the adit, that is, about 550 feet below the outcrop.

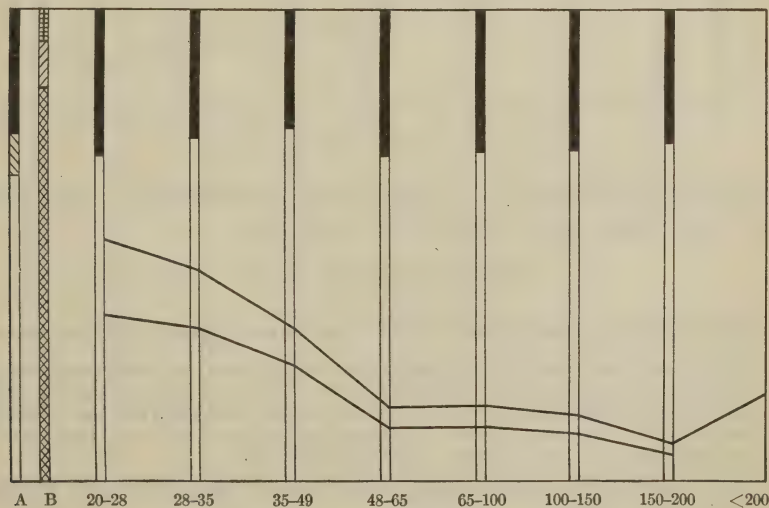


FIGURE 44.—Results of tests of sample 36.

The workable part of the bed was the lower  $3\frac{1}{2}$  feet, which was overlain by a thin layer of shale and 8 to 9 inches of low-grade ore discarded in mining.

## COMMENTS ON TESTS OF ORE FROM IRONDALE SEAM.

Sample 37 is slightly better quality than the average ore of the lower bench of the Big seam. All three samples doubtless represent hard ore, and their average composition is 38 per cent Fe and 22.1 per cent  $\text{SiO}_2$ , or about the same as that of the lower bench Big seam ore. The concentrates average 45.1 per cent Fe and 14.2 per cent  $\text{SiO}_2$ , and hence are the same in iron, but 1.5 per cent higher in silica than the concentrates from the lower bench of the Big seam. A product composed of concentrates plus slimes show the same difference. However, the ratio of concentration averages a little less for the Irondale ores, so that the results obtained are practically the same as those obtained with the lower bench Big seam ores. Hence

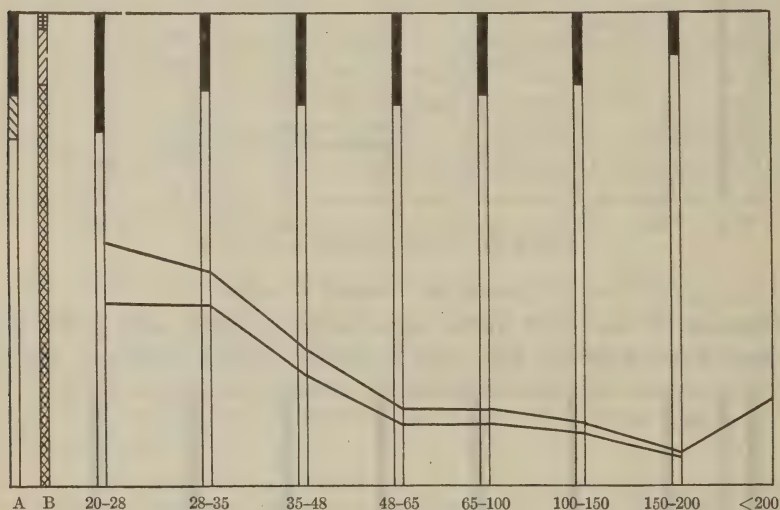


FIGURE 45.—Results of tests of sample 37.

the tests indicate that the Irondale ores are amenable to any treatment that might prove feasible for those ores.

## ORE FROM THE IDA SEAM.

The following description of the Ida seam is given by Burchard:<sup>a</sup>

This bed consists of 2 to 6 feet of rather siliceous ore associated with 14 to 16 feet of ferruginous sandstone. \* \* \* Where worked, the bed is 3 to 5 feet thick, and soft ore only has been obtained from it in surface workings. Such ore carries 35 to 44 per cent of metallic iron, with a corresponding range in silica of 42 to 32 per cent.

Two samples from this seam were tested, one from the Ruffner mine and the other from the Hammond.

<sup>a</sup> Burchard, E. F., and Butts, Charles, Iron ores, fuels, and fluxes of the Birmingham district, Alabama, with chapters on the origin of the ores, by E. C. Eckel: U. S. Geol. Survey Bull. 400, 1910, p. 46.

## SAMPLE 38. RUFFNER MINE.

The outcrop of the Ida seam at the Ruffner mine is 5 feet thick, and the sample was taken at the face of an old open cut in what Mr. C. E. Barrett, the superintendent, called hard ore.

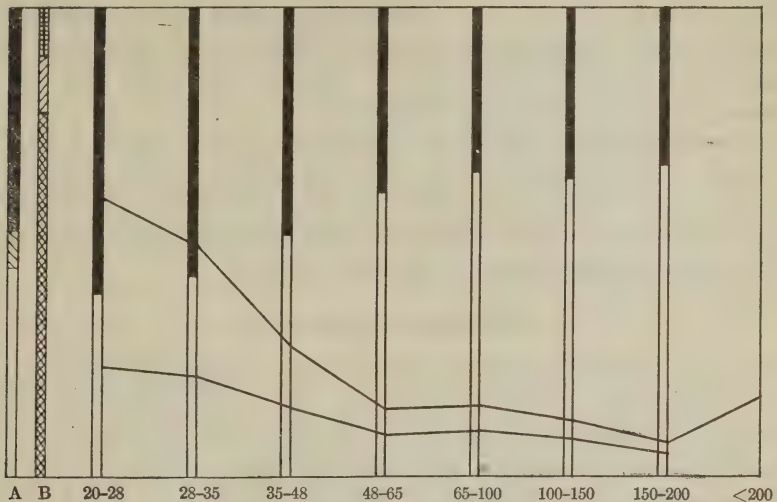


FIGURE 46.—Results of tests of sample 38.

## SAMPLE 39. HAMMOND MINE.

Sample 39 was taken from the outcrop of the Ida seam along the face of the bed at the outside incline.

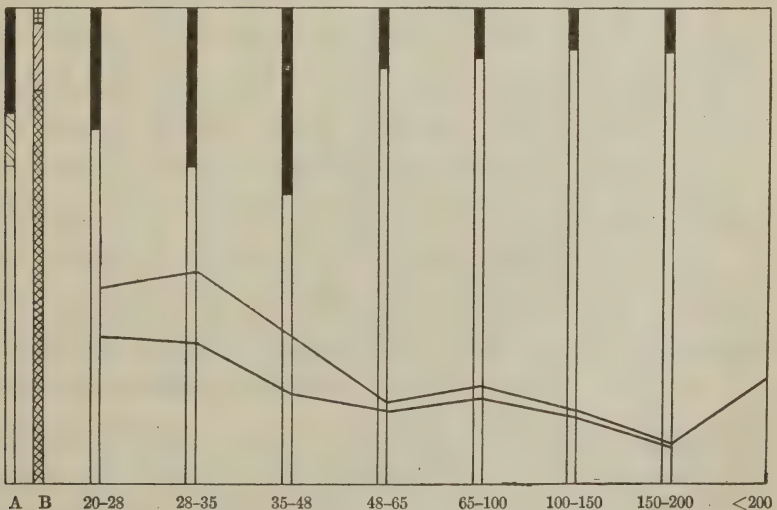


FIGURE 47.—Results of tests of sample 39.

## COMMENTS ON TESTS OF ORE.

One sample represents hard ore, the other soft ore. The sample of soft ore is plainly finer grained than the sample of hard ore,

as is shown by the screen analysis curves (figs. 46 and 47). The sample of hard ore (sample 38) is the poorest ore tested, carrying less than 30 per cent iron and more than 40 per cent silica, yet it yielded a concentrate carrying 47.5 per cent Fe and 14.7 per cent  $\text{SiO}_2$ , or just 2 per cent more iron and 2 per cent less silica than the average of the concentrates from the lower bench Big seam ore. This result would indicate that the siliceous Ida seam ores are as amenable to concentration as that ore.

The crude sample of soft ore (sample 39) runs high in iron, but its silica content is prohibitive. The concentrates from this sample show the high iron content obtained only in soft ore concentrates, and a silica content well within the allowable limit, and hence they can be advantageously used in the blast furnace.

### RÉSUMÉ OF RESULTS.

There is a steady market in the Birmingham district for two types of iron ores—the calcareous hard red ores and the siliceous brown ores. The average composition of the hard red ores, as given by Phillips,<sup>a</sup> is 37 per cent Fe, 13.44 per cent  $\text{SiO}_2$ , 16.20 per cent CaO. The average of 23 samples of washed brown ore, as given by Burchard,<sup>b</sup> is 45 per cent Fe, 14.6 per cent  $\text{SiO}_2$ , 0.74 per cent Mn. Also there is available a limited quantity of workable soft ore, having an average composition, according to Phillips,<sup>c</sup> of 50.80 per cent Fe, 18.50 per cent  $\text{SiO}_2$ , and 1.20 per cent CaO. All of these ores average from 3 to 4 per cent  $\text{Al}_2\text{O}_3$ . As regards their smelting, the soft ores are in the same class as the brown ores, a higher iron content being offset by a higher silica content. The hard red ores grade by decrease of lime and increase of silica into siliceous ores, which, on account of their low iron content, can not compete with the brown ores, and hence are unworkable at present. Under favorable conditions, hard red ores of the average iron content can not be worked profitably if they carry about 8 per cent more silica than lime. By far the greater part of the red ore of the district falls below these requirements, and hence requires concentration to become available. As the concentrates obtained in the tests described herein are themselves siliceous and not self-fluxing, the brown ores serve as a basis for determining the value of the product obtained by concentration. Such a comparison follows:

Six samples of Big seam upper-bench ore taken underground were tested. Sample 4 from the Hammond mine, close to the surface, represents soft ore and includes the lower bench. Its concentrates

<sup>a</sup> Phillips, W. B., *Iron making in Alabama*, 3d ed.: Geol. Survey of Alabama, 1912, p. 35.

<sup>b</sup> Burchard, E. F., and Butts, Charles, *Iron ores, fuels, and fluxes of the Birmingham district, Alabama*: U. S. Geol. Survey Bull. 400, 1910, p. 169.

<sup>c</sup> Phillips, W. B., work cited, p. 30.



run 51.7 per cent Fe and 18.4 per cent  $\text{SiO}_2$ ; in other words, they equal the average workable soft ore, and their iron content is more than enough to carry the 4 per cent excess in silica as compared with the average brown ore. The crude ore is unworkable, with silica in excess of iron.

Sample 1 from the Ishkooda mine is a workable, self-fluxing ore and was tested in order to ascertain the results of concentration. The concentrate is self-fluxing and 10 per cent higher in iron.

Sample 2 and 3 from the Spaulding mine represent upper bench ores that are workable only when iron is in good demand. Their concentrates are higher in iron than the average brown ores and, being nearly self-fluxing, are superior to those ores.

Two samples from the Ruffner mine, samples 5 and 6, represent extremely poor ore, with silica in excess of iron, averaging 28.8 per cent Fe and 30.9 per cent  $\text{SiO}_2$ . The concentrates are of much better quality, averaging 40.5 per cent Fe and 17.1 per cent  $\text{SiO}_2$ , but are too poor to be marketable.

These experiments show that the unworkable upper bench hematites of the Big seam will yield a concentrate of greater value than the brown ores unless the grade of the crude hematites is very low.

Four samples of ore were taken from the outcrop of the Big seam at the Frank White open cut. Two of these, samples 7 and 8, represent the upper 6 feet of the seam (all that was worked) but are so much lower in iron and higher in silica than the average soft ore that this part of the bed can be worked only when the iron trade is unusually active. Concentration yielded a high-grade product, carrying 53 per cent Fe and 12.9 per cent  $\text{SiO}_2$ . The middle 6 feet, sample 9, is of very low grade, with 5 per cent more silica than iron, yet the concentrate is only a little poorer than the average soft ore mined. Sample 10, from the lower 6 feet, shows about the same composition as the ore from the upper 6 feet, but the concentrate was poorer, having about the composition of the average workable soft ore, probably because of the finer grain. The entire seam averages 42.2 per cent Fe and 30.5 per cent  $\text{SiO}_2$ , and the concentrates from it 50.9 per cent Fe and 17.2 per cent  $\text{SiO}_2$ . Therefore, concentration at this mine could make a product of slightly better grade than the average of the soft ores out of an 18-foot seam of ore, of which only the upper 6 feet can even be considered as available in its crude state.

Twenty-four samples of Big seam lower bench ore were tested, of which one, sample 11 from the Ruffner mine, represents an extremely low-grade ore from the northeastern part of the district. The others represent the lower bench in that part of the district, where the upper bench is worked extensively and the lower bench is left unmined.

The sample of lower bench Ruffner ore is slightly poorer than the two samples of the upper bench ore tested, but the concentrate runs a little better. However, the concentrate is below the grade of the brown ores, and hence concentration would probably be unprofitable.

The 23 samples from Red Mountain between the Spaulding and Fossil mines show considerable uniformity in composition and in the results of concentration. This fact tends to emphasize the conclusions that can be based on the tests. The samples averaged 37.3 per cent Fe, 23.6 per cent  $\text{SiO}_2$ , and 8.9 per cent  $\text{CaO}$ ; that is, their iron content is about the same as the average of the workable upper bench ores, but their silica content is 10 per cent higher and their lime content is more than 7 per cent lower. The concentrates obtained from these samples contain 0.4 per cent more iron and 2 per cent less silica than the brown ores, and, in addition, have a lime content of 8.1 per cent. Thus concentration yields a product of greater value than the brown ores. These results are the more encouraging because they were obtained with an iron recovery of 94.1 per cent and a concentration of 4 tons of crude ore into 3 tons of final product.

Three samples of ore from the Irondale seam were tested, samples 35 and 36 from the Ruffner mine, and sample 37 from the Hammond mine. The average iron and silica contents of these ores, 38 per cent Fe and 22.1 per cent  $\text{SiO}_2$ , is about the same as those of the lower bench Big seam ores. The concentrates average 45.1 per cent Fe and 14.2 per cent  $\text{SiO}_2$ , a little higher in silica than the Big seam lower bench concentrates, but still as good as the brown ores. In addition the concentrates doubtless have an appreciable lime content that would increase their value. On the whole, then, the ores tested from the Irondale seam have about the same composition as the Big seam lower bench ores from the producing part of the district, and are approximately as amenable to concentration.

Two samples of Ida seam ore were tested, one being hard ore and the other soft. Sample 37 from the Ruffner mine represents a poor hard ore, carrying 28.2 per cent Fe and 40.8 per cent  $\text{SiO}_2$ . The slimes from the test are so low grade that mixing them with the concentrates appreciably lowers the grade of the product; discarding them reduces the iron recovery to only 77.8 per cent, but the concentrate contains 47.5 per cent Fe and 14.7 per cent  $\text{SiO}_2$ , and is as low in silica as the average brown ore, with 2.5 per cent more iron. These results indicate that the ore has undergone partial alteration and is to be classed as semihard. The soft ore, sample 39 from the Hammond mine, is too poor to market in the crude state, but its concentrate had the same silica content as the brown ores and 10 per cent more iron.

With but few exceptions, then, and these being the unusually low-grade material, the experiments in concentrating the siliceous red

ores have yielded final products that would have a value in excess of that possessed by the brown ores of the district.

The experiments above are based on crushing the ore to pass 20 mesh, probably as fine as would be practical. In order to determine, however, the degree to which the grade of the concentrate could be raised by still finer crushing, in five tests part of the ore was crushed to pass 35 mesh, and the concentrate resulting from the separation of both sizes was determined. For purposes of comparison the results are tabulated below.

*Results of finer crushing.*

Sample.	20-mesh test.					35-mesh test.					Slimes.	
	Concentrates.	Slimes.	Tailings.	Iron.	Silica.	Concentrates.	Slimes.	Tailings.	Iron.	Silica.	Iron.	Silica.
	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>
6.....	61.0	10.9	28.1	42.5	20.6	41.0	19.8	39.2	43.7	16.7	34.5	13.0
14.....	63.3	9.2	27.5	45.1	13.5	52.7	21.4	25.9	45.3	13.0	44.3	14.9
20.....	60.4	9.4	30.2	42.3	16.1	47.6	21.0	31.4	42.1	13.3	39.8	12.4
27.....	68.2	10.0	21.8	43.5	13.2	55.8	21.1	23.1	45.7	11.0	44.0	12.8
29.....	71.5	11.3	17.2	48.0	12.1	58.4	22.5	19.1	48.1	11.7	43.8	16.6
Average..	64.9	10.2	25.0	44.3	15.1	51.1	21.2	27.7	45.0	13.1	.....	.....

These results show a very small average increase in iron in the finer concentrates and a 2 per cent decrease in silica. The amount of the tailings increased only a little, the change being almost entirely in the amount of the slimes at the expense of the concentrates. In the table the first and last slime analyses are of the 35-mesh slimes and the other three of the 20-mesh slimes. Thus increased sliming lowers the average grade of the slimes. If the slimes and concentrates are mixed to form a final product with as high iron recovery as for the similar product from the 20-mesh crushing, the grade of it will be but little if any higher and there will be no increase in grade of the final product to compensate for the finer crushing. Hence, 20-mesh crushing is fine enough to produce an adequate freeing of the constituent ore particles to yield an efficient separation.

In the above summary, except where otherwise indicated, the final product of concentration is considered to be the combined concentrate and slimes. The average composition of all the concentrates is 46.4 per cent Fe and 13.4 per cent  $\text{SiO}_2$ , and that of all the slimes is 44.7 per cent Fe and 13.5 per cent  $\text{SiO}_2$ . Of the total iron in the crude ore, that in the concentrates averages 80.6 per cent and that in the slimes 13.1 per cent. The average quantity of concentrates is 63.3 per cent, and that of slimes is 10.6 per cent. Consequently, if the slimes are added to the concentrates, the average composition of the final product will be 46.2 per cent Fe and 13.4 per cent  $\text{SiO}_2$ ; in other words, this procedure does not change the silica content



and lowers the iron content only 0.2 per cent, but the average iron recovery is raised 13.1 per cent, or from 80.6 per cent to 93.7 per cent. The advantages of mixing these two products in raising the iron recovery and eliminating the need of slime treatment are obvious.

That crushing alone brings about a concentration of the iron in the finer sizes and a concentration of the silica in the coarser sizes is strikingly shown, the slimes (less than 200 mesh) from the crude ore being of practically the same composition as the concentrates from the sizes coarser than 200 mesh. In order to show this tendency in more detail the concentrates and tailings of each mesh size of five samples were analyzed, and from these analyses the composition of the crude ore of each of these sizes was calculated. The results are tabulated below.

*Increase of iron in the slimes.*

	Sample 3.		Sample 8.		Sample 10.		Sample 19.		Sample 39.		Average.	
	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .	Fe.	SiO <sub>2</sub> .
	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>
Crude ore.....	32.4	29.3	35.9	27.0	35.7	15.0	38.4	20.2	39.5	19.3	36.4	22.2
20 to 28.....	28.8	40.1	29.4	38.5	32.5	20.5	32.1	32.3	31.9	29.9	30.9	32.3
28 to 35.....	29.6	33.1	32.2	33.0	36.1	15.9	36.6	23.8	35.2	24.4	33.9	26.0
35 to 48.....	31.8	24.7	36.1	24.0	37.1	14.0	37.4	19.7	37.3	19.9	35.9	20.5
48 to 65.....	32.5	20.1	39.5	19.3	35.0	12.4	38.6	16.0	38.3	17.3	36.8	17.0
65 to 100.....	33.9	18.3	40.9	18.7	35.3	12.2	39.9	14.8	38.7	16.2	37.7	16.0
100 to 150.....	36.3	16.7	43.5	16.4	38.4	10.9	42.3	13.2	43.0	15.2	40.7	14.5
150 to 200.....	43.3	12.7	45.1	16.0	42.3	9.7	44.7	13.3	45.3	14.6	44.1	13.3
200.....	39.8	12.7	46.1	14.0	40.0	8.9	45.5	11.3	47.4	11.7	43.8	11.7
Concentrate....	42.3	16.1	45.5	11.9	46.2	7.9	45.3	12.0	45.3	11.8	44.9	11.9

This table shows that the two coarsest sizes, 20 to 28 mesh and 28 to 35 mesh, are of lower grade than the crude ore, whereas the finer sizes are of increasingly higher grade.

**PRACTICAL VALUE OF THE EXPERIMENTS.**

The bearing of the results of these experiments on probable results of concentration on a commercial scale by wet methods or with magnetic separation will now be considered. The author knows that the method of concentration he used is not feasible on a commercial scale, and its efficiency of separation is not obtainable in a commercial plant. The experiments were made to determine the degree to which the silica might be separated from the iron—that is, granted a perfect separation of the mineral particles, what is the result?

If this result indicates that concentration is feasible, the devising of a commercial method of separation would be encouraged. If, on the other hand, a perfect separation of the mineral particles does not yield a marketable product of sufficient value, there is no use working at separation on a commercial scale. The results obtained in the



experiments have demonstrated that, so far as the experiments go, a solution of the problem is possible.

Results obtained in the application of the Moxham-du Pont haloid process to the concentration of low-grade iron ores open the possibility of the commercial application of the method used in this investigation and may render the results of these experiments directly applicable. But while this process is in the experimental stage, it will be better to consider the bearing of these results on separation by ordinary wet methods or by magnetic methods.

Wet methods of separation are based essentially on the differences in specific gravity of the mineral components of an ore. In the author's experiments the ores were crushed to pass 20 mesh, a fineness that is not at all unusual in ore mills. As the iron oxide in the ore tested is much softer than the silica that is to be removed, crushing causes an increase of iron in the finer sizes. So marked is this concentration of the iron that the crude material passing through 200 mesh has practically the same iron content as that obtained by concentrating the material coarser than 200 mesh. This fact is of great importance as it eliminates the slime problem in the use of wet methods. The material finer than 200 mesh may be separated by washing immediately after crushing, allowed to settle, and then added to the concentrate obtained from the rest of the material. The concentrates obtained with the same ratio of concentration will not be of as high grade as those made in the experiments because of less complete separation, but this difficulty could be met by increasing the ratio of concentration a little, thus raising the grade of the concentrate and lowering the iron recovery. As the concentrates from the tests of lower bench ores of the Big seam, for example, represented an iron recovery of 94.1 per cent, it is evident that a little lowering of the iron recovery to meet the decreased efficiency of commercial work would still permit a good recovery and give a concentrate of high enough grade. Hence there seems to be no reason to doubt the possibility of obtaining equally good concentrates, although with a lower iron recovery, on a commercial scale by wet methods.

Equally favorable conclusions can be drawn in regard to the results that may reasonably be expected from applying magnetic separation to remove the more magnetic particles of the crushed ore, as in an ore similar to that under discussion the more magnetic particles are those high in iron and low in silica. In other words, the magnetic susceptibility is determined by the same factors as the specific gravity and is a linear function of the latter. Consequently, crushing frees the ore particles for magnetic separation to exactly the same degree as for specific gravity separation. Therefore a perfect magnetic separation should yield results identical with those obtained by a

perfect specific gravity separation. But in ordinary practice a perfect magnetic separation is no more obtainable than a perfect gravity separation. In either case the grade of the final product can be maintained by increasing the ratio of concentration a little at the expense of the iron recovery. As stated above, in the author's concentration experiments iron recovery was so high that a small decrease in it would not be prohibitive.

The experiments have, consequently, demonstrated the possibility of producing from ores now valueless and being permanently lost by present mining methods a marketable product. Also, it is probable that as good results can be obtained commercially by any of the usual methods of ore treatment if the decrease of efficiency is compensated by increasing the ratio of concentration.

#### COMMERCIAL ASPECTS OF THE PROBLEM.

As the tests indicate that concentrates of higher grade than the best of the ores now available in the Birmingham district can be obtained from the siliceous ores of the district, the question remains whether concentration would be profitable. The answer requires the determination of two things—first, the cost of crushing, concentrating, and nodulizing the concentrates; second, the value of the concentrates as compared with the crude ore. The author does not have the necessary data and does not consider himself competent to supply such figures, for they should be based on experience with concentration on a large scale and on familiarity with mining costs and the price of ore in the district. However, some general suggestions are offered.

Beneficiation of low-grade iron ores is widely practiced; it includes magnetic separation and nodulizing of low-grade magnetites. Magnetic concentration of the siliceous red hematites would involve higher treatment cost than for magnetites because, generally speaking, finer crushing would be needed. On the other hand, these ores can be mined cheaper than many magnetites thus treated. Gravity methods of wet concentration for low-grade iron ores have been successfully used on a large scale in the Lake Superior region, and have made available great quantities of ore that are too low in grade to be shipped crude. Hence it is reasonable to presume that the costs of treatment would not necessarily be prohibitive.

The margin allowable for treatment costs can be determined more nearly. During the years 1910 to 1915 the red ores and the brown ores of the district showed the following range in prices, according to the United States Geological Survey:<sup>a</sup>

---

<sup>a</sup> Figures for 1910 and 1911 from Mineral Resources of the United States for 1911, Part 1, p. 130. Figures for 1912 and 1913 from Mineral Resources of the United States for 1913, Part I, p. 305. Figures for 1914 and 1915 are from Mineral Resources of the United States for 1915, Part 1, p. 291.

*Prices of red and brown ores of Birmingham district.*

	1910	1911	1912	1913	1914	1915	Average.
Red ores.....	\$1.19	\$1.17	\$1.16	\$1.18	\$1.21	\$1.25	\$1.19
Brown ores.....	1.53	1.47	1.43	1.61	1.49	1.68	1.54

During these six years the average price of the brown ores was 35 cents in excess of that of the red ores. It has been seen that the value of the concentrates would exceed that of the brown ores, and the cost of mining the lower bench ores in mines working the upper bench ought to be less than the present cost of mining the upper bench alone. Therefore, the margin allowable for treatment should be in excess of 35 cents. How much larger it might be can be determined by those having complete data on mining costs and the value of the ores as determined by chemical composition.

That one must guard against assuming too low a cost for mining lower bench as compared with upper bench ores is pointed out by Burchard, who says:<sup>a</sup>

To recover the lower ore at present would, however, introduce serious complications in mining. At the Tennessee company's mines the slopes are cut in the lower bench of the seam about 8 feet below the working level of the headings, so that tramcars of ore may be dumped over the tippie into the skip in the slope. If the lower bench of ore were mined also, the slopes would have to be depressed 8 feet below the bottom of this bench in order to accommodate skip haulage. This would necessitate maintaining for some distance a slope height of 25 to 28 feet, which would be excessive and would involve deepening and retimbering the present slopes. Beyond the present faces the slopes might be driven below the parting, leaving the upper bench above the slope. The matter of setting props to support the roof in the workings where 18 to 20 feet of ore are taken out would also prove inconvenient.

In slopes where tram haulage is employed the problem is simpler, since it is not necessary to depress the slope below the bottom of the ore that is being mined. Part of the ore from the lower bench can be obtained in either case by cutting the floors of the rooms lower. By this method, however, the floors become too flat to permit the ore to be shot down to the cars, and the expense of moving it is thereby increased.

Evidently the problem is discouraging in view of present costs and conditions, but if for every ton of ore mined from the lower bench nearly a ton of ore in the upper bench might be saved for future use, it would seem worth while to carry the investigation as far as possible, even to the extent of equipping one slope to mine the whole seam on a scale that would enable the relative costs to be compared on a working basis.

In spite of such difficulties there certainly ought to be a margin in favor of the lower bench ore. As those ores are not utilized at

<sup>a</sup> Burchard, E. F., and Butts, Charles, Iron ores, fuels, and fluxes of the Birmingham district, Ala.: U. S. Geol. Survey Bull. 400, 1910, pp. 124, 125.



present and are being destroyed as reserves, and as the mines are now equipped for mining the upper bench ore, none of the charges now assumed by the upper bench ore should be distributed between it and the cost of producing the lower bench ore. The cost of mining both ores should be determined, and then from that cost should be subtracted the present cost of mining the upper bench ore. The difference is all that should fairly be charged against the lower bench costs and would certainly be less than the present figures for the upper bench ores. The difference would be applicable to meeting concentration charges. Moreover, a decidedly important consideration would be the doubling of the ore reserves.

From this consideration of the factors involved, it seems reasonable to assume that the margin allowable for meeting concentration charges is considerably more than 35 cents.



## PUBLICATIONS ON MINERAL TECHNOLOGY.

A limited supply of the following publications of the Bureau of Mines has been printed and is available for free distribution until the edition is exhausted. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

The Bureau of Mines issues a list showing all its publications available for free distribution as well as those obtainable only from the Superintendent of Documents, Government Printing Office, on payment of the price of printing. Interested persons should apply to the Director, Bureau of Mines, for a copy of the latest list.

### PUBLICATIONS AVAILABLE FOR FREE DISTRIBUTION.

BULLETIN 64. The titaniferous iron ores in the United States; their composition and economic value, by J. T. Singewald, jr. 1913. 145 pp., 16 pls., 3 figs.

BULLETIN 70. A preliminary report on uranium, radium, and vanadium, by R. B. Moore and K. L. Kithil. 1913. 100 pp., 2 pls., 2 figs.

BULLETIN 73. Brass furnace practice in the United States, by H. W. Gillett. 1914. 298 pp., 2 pls., 23 figs.

BULLETIN 84. Metallurgical smoke, by C. H. Fulton. 1915. 94 pp., 6 pls., 15 figs.

BULLETIN 85. Analyses of mine and car samples of coal collected in the fiscal years 1911 to 1913, by A. C. Fieldner, H. I. Smith, A. H. Fay, and Samuel Sanford. 1914. 444 pp., 2 figs.

BULLETIN 92. Feldspars of the New England and Northern Appalachian States, by A. S. Watts. 1916. 180 pp., 3 pls., 22 figs.

BULLETIN 103. Mining and concentration of carnotite ores, by K. T. Kithil and J. A. Davis. 1917. 89 pp., 14 pls., 5 figs.

BULLETIN 104. Extraction and recovery of radium, uranium, and vanadium from carnotite, by C. L. Parsons, R. B. Moore, S. C. Lind, and O. C. Schaefer. 1915. 124 pp., 14 pls., 9 figs.

BULLETIN 106. The technology of marble quarrying, by Oliver Bowles. 1916. 174 pp., 12 pls., 33 figs.

BULLETIN 124. Sandstone quarrying in the United States, by Oliver Bowles. 1917. 143 pp., 6 pls., 19 figs.

BULLETIN 128. Refining and utilization of Georgia kaolins, by I. E. Sproat. 1916. 55 pp., 5 pls., 11 figs.

BULLETIN 146. Technology of salt making in the United States, by W. C. Phalen. 1917. 149 pp., 24 pls., 10 figs.

TECHNICAL PAPER 8. Methods of analyzing coal and coke, by F. M. Stanton and A. C. Fieldner. 1913. 42 pp., 12 figs.

TECHNICAL PAPER 50. Metallurgical coke, by A. W. Belden. 1913. 48 pp., 1 pl., 23 figs.

TECHNICAL PAPER 76. Notes on the sampling and analysis of coal, by A. C. Fieldner. 1914. 59 pp., 6 figs.

TECHNICAL PAPER 95. Mining and milling of lead and zinc ores in the Wisconsin district, Wisconsin, by C. A. Wright. 1915. 39 pp., 2 pls., 5 figs.

TECHNICAL PAPER 102. Health conservation at steel mills, by J. A. Watkins. 1916. 36 pp.

TECHNICAL PAPER 110. Monazite, thorium, and mesothorium, by K. L. Kithil. 1915. 32 pp., 1 fig.

TECHNICAL PAPER 111. Safety in stone quarrying, by Oliver Bowles. 1915. 48 pp., 5 pls., 4 figs.

TECHNICAL PAPER 126. The casting of clay wares, by T. G. McDougal. 1916. 26 pp., 6 figs.

TECHNICAL PAPER 133. Directions for sampling coal for shipment or delivery, by G. S. Pope. 1917. 15 pp.

TECHNICAL PAPER 136. Safe practice at blast furnaces, by F. H. Willcox. 1916. 73 pp., 1 pl., 43 figs.

TECHNICAL PAPER 143. Ores of copper, lead, gold, and silver, by C. H. Fulton. 1916. 41 pp.

TECHNICAL PAPER 155. Gypsum products, their preparation and uses, by R. W. Stone. 1917. 67 pp., 9 pls., 10 figs.

TECHNICAL PAPER 177. Preparation of ferro-uranium, by H. W. Gillett and E. L. Mack. 1917. 46 pp., 2 figs.

#### PUBLICATIONS THAT MAY BE OBTAINED ONLY THROUGH THE SUPERINTENDENT OF DOCUMENTS.

BULLETIN 11. The purchase of coal by the Government under specifications, with analyses of coal delivered for the fiscal year 1908-9, by G. S. Pope. 80 pp. 10 cents.

BULLETIN 12. Apparatus and methods for the sampling and analysis of furnace gases, by J. C. W. Frazer and E. J. Hoffman. 1911. 22 pp., 6 figs. 5 cents.

BULLETIN 22. Analyses of coals in the United States, with descriptions of mine and field samples collected between July 1, 1904, and June 30, 1910, by N. W. Lord, with chapters by J. A. Holmes, F. M. Stanton, A. C. Fieldner, and Samuel Sanford. 1912. Part I, Analyses, pp. 1-321; Part II, Descriptions of samples, pp. 321-1129. 85 cents.

BULLETIN 29. The effect of oxygen in coal, by David White. 80 pp., 3 pls. 20 cents.

BULLETIN 41. Government coal purchases under specifications, with analyses for the fiscal year 1909-10, by G. S. Pope, with a chapter on the fuel-inspection laboratory of the Bureau of Mines, by J. D. Davis. 1912. 97 pp., 3 pls., 9 figs. 15 cents.

BULLETIN 42. The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs. 20 cents.

BULLETIN 47. Notes on mineral wastes, by C. L. Parsons. 1912. 44 pp. 5 cents.

BULLETIN 53. Mining and treatment of feldspar and kaolin in the Southern Appalachian region, by A. S. Watts. 1913. 170 pp., 16 pls., 12 figs. 35 cents.

BULLETIN 63. Sampling coal deliveries and types of Government specifications for the purchase of coal, by G. S. Pope. 1913. 68 pp., 4 pls., 3 figs. 10 cents.

BULLETIN 71. Fuller's earth, by C. L. Parsons. 1913. 38 pp. 5 cents.

BULLETIN 81. The smelting of copper ores in the electric furnace, by D. A. Lyon and R. M. Keeney. 1915. 80 pp., 6 figs. 10 cents.

BULLETIN 108. Melting aluminum chips, by H. W. Gillett and G. M. James. 1916. 88 pp. 10 cents.

BULLETIN 111. Molybdenum; its ores and their concentration, with a discussion of market, prices, and uses, by F. W. Horton. 1916. 132 pp., 18 pls., 2 figs. 30 cents.

BULLETIN 122. The principles and practice of sampling metallic metallurgical materials, with special reference to the sampling of copper bullion, by Edward Keller. 1916. 101 pp., 13 pls., 31 figs. 20 cents.

TECHNICAL PAPER 41. The mining and treatment of lead and zinc ores in the Joplin district, Mo., a preliminary report, by C. A. Wright. 1913. 43 pp., 5 figs. 5 cents.

TECHNICAL PAPER 60. The approximate melting points of some commercial copper alloys, by H. W. Gillett and A. B. Norton. 1913. 10 pp., 1 fig. 5 cents.

TECHNICAL PAPER 88. The radium-uranium ratio in carnotites, by S. C. Lind and C. F. Whittemore. 1915. 29 pp., 1 pl., 4 figs. 5 cents.

TECHNICAL PAPER 90. Metallurgical treatment of the low-grade and complex ores of Utah, a preliminary report, by D. A. Lyon, R. H. Bradford, S. S. Arentz, O. C. Ralston, and C. L. Larson. 1915. 40 pp. 5 cents.

TECHNICAL PAPER 93. Graphic studies of ultimate analyses of coals, by O. C. Ralston, with a preface by H. C. Porter. 1915. 41 pp., 3 pls., 6 figs. 10 cents.

TECHNICAL PAPER 109. Composition of the natural gas used in 25 cities, with a discussion of the properties of natural gas, by G. A. Burrell and G. G. Oberfell. 1915. 22 pp. 5 cents.





# INDEX.

A.		Page.			Page.
Alabama, iron ores in	-----	8-9	Concentrates, ore, analyses of	41, 52,	
Alumina, percentage of, in				58, 65, 69, 72	
brown ore	-----	57	Composition of	-----	33, 34,
In hematite ores	9-10, 17, 76			35, 36, 37, 38, 39	
Analyses, average, of brown			Self-fluxing character of	-----	13
ores	56-57, 72, 76		Concentration of hematite ores,		
Of concentrates	41, 52, 58,		profit in	-----	82, 83
	65, 69, 72		Ratio of	-----	12, 13, 33, 39, 45
Of red hematite ores	-----	10	Concentration tests, methods		
	17, 41, 52, 72		of	-----	13, 14-15
Of tailings	41, 52, 58, 65, 69, 72		Results of	-----	12, 13, 18-39, 76-80
			Plotting of	-----	15-16
			<i>See also</i> names of mines.		
B.			E.		
Bessemer, Ala., ore near, compo-					
sition of	-----	17	Eckel, E. C., work of	-----	8
Big seam, description of	10-11, 16		F.		
Ore in, character of	-----	17			
Ore from, average analyses			Fossil mine ore, composition of	38, 69	
of	10, 17, 72		Concentration tests, results		
Concentrates from, value			of	29, 30, 31, 38	
of	77		Discussion of	66-67,	
Concentration tests of	45-72			69, 70-71	
Comments on	71-72		Frank White mine ore, compo-		
Crushing of, effect of	71		sition of	34, 35	
Description of	45-46		Concentration tests, results		
Brown ores, average analyses			of	20-22, 34, 35	
of	56-57, 72, 76		Discussion of	49-50	
Comparison with red ores	61, 66-		Character of	46	
	67, 76, 78, 79		H.		
Composition of	73				
Prices of	83		Haloid process of concentration	13, 81	
Burchard, E. R., work of	8, 13, 16		Hammond mine ore, character		
Butts, Charles, work of	7		of	42, 46, 75-77	
C.			Composition of	34, 39	
Carbonic acid, percentage of, in			Concentration tests, results		
"hard ore"	-----	17	of	19, 32, 33, 34, 39, 76-77	
Clark, W. B., acknowledgment			Discussion of	46	
to	-----	8	Notes on	73	
Clinton ores, treatment of	11, 14				

Hematite ores, crude, composition of -----	Page.	- L.	Page.
12, 13, 52, 58, 65, 72		Leibensperger, Raymond, work of -----	8
Concentration tests, results of -----	12	Lime, percentage of, in concentrates -----	12, 33,
<i>See also</i> mines named.		34, 35, 36, 37, 38, 39, 46, 72	
Hickory Nut seam, character and thickness of -----	9	In crude ore -----	12, 33, 46, 72
		In red hematite ores -----	9, 12,
I.		17, 66, 76, 78	
Ida seam, description of -----	74-75	Low-grade iron ores, testing of -----	13, 14
Ore from, character of -----	75-76		
Iron, percentage of, in brown ores -----	57, 76	M.	
In concentrates -----	12,	Magnesia, percentage of, in hematite ores -----	10
13, 41, 43, 45, 49, 50, 51,		In brown ore -----	76
52, 54, 57, 58, 59, 64, 65,		Moxham-du Pont haloid process -----	13, 81
66, 69, 72, 76, 77, 78			
In crude samples -----	12,	N.	
33, 34, 35, 36, 37, 38, 39,		Nitze, H. B. C., work of -----	11
41, 49, 52, 58, 65, 69, 72			
In hematite ores -----	9, 10,	O.	
12, 13, 17, 45, 46, 49,		Ores. <i>See</i> Hematite ores; Iron ores.	
50, 52, 54, 56-57, 66,			
71, 72, 74, 76, 77		P.	
In slimes -----	33,	Penhallegon, W. J., acknowledgment to -----	8
34, 35, 36, 37, 38, 39, 80		Phillips, W. B., work of -----	8, 11,
In tailings -----	13,	12, 13, 17	
33, 34, 35, 36, 37, 38,		Phosphorus, percentage of, in hematite ores -----	10, 17
39, 41, 52, 58, 65, 69		Pleasant Hill, Ala., ore from, concentration tests of, results of -----	13-14
Recovery of, in concentration tests -----	41,		
42, 45, 46, 49, 51, 52, 54,		R.	
56, 57, 58, 59, 61, 64, 66,		Red ores. <i>See</i> Hematite ores.	
67, 71, 72, 76, 77, 79, 80		Rich Patch mines, Va., testing of limonites at -----	14
Iron ores, Birmingham district, geology of -----	9	Ruffner mine ore, character of -----	43, 50
Thickness of -----	9	Composition of -----	34, 35, 39
Brown, average analyses of -----	56-57, 72, 76	Concentration tests, results of -----	19, 20, 22,
Low-grade, concentration of -----	13, 14	32, 33, 34, 35, 39, 50-52	
<i>See also</i> Hematite ores; mines named.		Discussion of -----	43-45,
Irondale seam ore, average analyses of -----	74	46, 50-52, 77, 78	
Description of -----	72-73	Notes on -----	43, 50, 72, 73, 75
Concentration tests of -----	74		
Ishkooda mine ore, composition of -----	33, 36		
Concentration tests of -----	18,		
25, 26, 27, 33,			
36, 45, 57-61, 77			
Notes on -----	40, 57-58, 60		

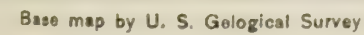
S.	Page.		Page.
Screening tests, method of	15-16	Spaulding mine ore, composition	
Plotting of	14-15	of	33, 34, 35-36
Results of	18-33	Concentration tests of, re-	
Self-fluxing, definition	17	sults of	18,
Silica in hematite ores	9, 10, 17	23-25, 33, 34, 35-36	
Percentage of, in brown ores	57, 76	Discussion of	40-42,
In concentrates	12,	45-46, 52, 53-54, 56-57	
33-39, 41, 43, 45, 46, 49, 50,		Notes on	40, 53, 55, 56
51, 52, 54, 57, 58, 59, 61, 64,		Sulphur, percentage of, in hema-	
65, 66, 69, 71, 72, 74, 76, 77, 78		tites	17
In red hematite ores	17, 45,	In iron ores	10
46, 49, 50, 74, 76, 77, 78		T.	
In tailings	33, 34,	Tailings, analyses of	41, 52,
35, 36, 37, 38, 39, 41, 52, 58, 69		58, 65, 69, 72	
Slimes, composition of	33, 34,	Composition of	33, 34,
35, 36, 37, 38, 39		35, 36, 37, 38, 39	
Iron in, percentage of	80	Percentage of	18-33, 79
Songo mine ores, composition of	37	W.	
Concentration tests of, re-		Water, percentage of, in "hard	
sults of	27-29, 37	ore "	17
Discussion of	64, 65-66	Wilkins, H. A. J., work of	11
Notes on	61-63, 64	Woodbridge, D. E., work of	7











*A. Hoen & Co Lith. Baltimore, Md*



DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

MOLYBDENUM; ITS ORES AND THEIR  
CONCENTRATION

WITH A DISCUSSION OF

MARKETS, PRICES, AND USES

BY

FREDERICK W. HORTON



WASHINGTON  
GOVERNMENT PRINTING OFFICE

1916

The Bureau of Mines, in carrying out one of the provisions of its organic act—to disseminate information concerning investigations made—prints a limited free edition of each of its publications.

When this edition is exhausted, copies may be obtained at cost price only through the Superintendent of Documents, Government Printing Office, Washington, D. C.

The Superintendent of Documents *is not an official of the Bureau of Mines*. His is an entirely separate office, and he should be addressed:

SUPERINTENDENT OF DOCUMENTS,  
*Government Printing Office,*  
*Washington, D. C.*

The general law under which publications are distributed prohibits the giving of more than one copy of a publication to one person. The price of this publication is 30 cents.

*First edition. October, 1916.*



# CONTENTS.

---

	Page.
Introduction.....	3
Scope and purpose of investigation.....	4
Present condition of the molybdenum industry.....	4
The future of the molybdenum industry.....	5
Acknowledgments.....	5
Part I. The molybdenum minerals.....	7
Molybdenite.....	7
Tests for molybdenite.....	8
Occurrences of molybdenite.....	8
Accompanying minerals.....	9
Jordisite.....	9
Wulfenite.....	9
Tests.....	10
Occurrence and accompanying minerals.....	10
Molybdlite.....	11
Tests for molybdlite.....	12
Occurrence and accompanying minerals.....	12
The rare molybdenum minerals.....	12
Ilsemaninite.....	12
Belonesite.....	13
Tests.....	13
Powellite.....	13
Tests.....	14
Occurrence.....	14
Pateraite.....	14
Achrematite.....	14
Tests.....	14
Eosite.....	14
Tests.....	15
A new molybdenum mineral.....	15
Doubtful species.....	16
Physical properties of molybdenum.....	17
Chemical properties of molybdenum.....	19
Uses of molybdenum.....	20
Molybdenum steels.....	21
Manufacture of molybdenum steels.....	22
Use of molybdenum in tool steels.....	22
Use of molybdenum in magnet steel.....	25
Use of molybdenum in acid-resisting steels.....	25
Use of molybdenum in other alloy steels.....	25
Ferromolybdenum.....	26
Use of molybdenum in stellite.....	26
Uses of metallic molybdenum.....	27
Use of molybdenum in chemicals.....	28
Production of molybdenum ores.....	29
Queensland.....	29
New South Wales.....	30
Norway.....	31

## Part I. The molybdenum minerals—Continued.

## Production of molybdenum ores—Continued.

	Page.
United States.....	32
Canada.....	34
Imports of molybdenum.....	34
Market for molybdenum.....	35
How molybdenum concentrates are marketed.....	36
Prices of molybdenite and of wulfenite concentrates.....	37
Qualitative tests for molybdenum.....	40
Quantitative determination of molybdenum.....	41
Solution.....	41
Precipitation.....	42
Conversion of precipitate to weighable form.....	43
Volumetric determination of precipitated molybdenum trisulphide.....	43
Method adopted by the Bureau of Mines.....	44

## Part II. Description of deposits.

General distribution of deposits.....	45
Arizona.....	45
Wulfenite at the Mammoth and Collins mines, Pinal County.....	46
Wulfenite at the Old Yuma mine, Pima County.....	48
Other occurrences of wulfenite in Arizona.....	50
Occurrence of molybdenite at the Leviathan mines, Mohave County.....	52
Molybdenite at the Smith & Sawyer and the Miller claims, Mohave County.....	55
Molybdenite property of Arizona Molybdenum Co., Pima County.....	55
Molybdenite at Leader mine, Pima County.....	56
Other occurrences of molybdenite.....	57
California.....	58
Molybdenite at property of Santa Maria Molybdenum Mining & Milling Co., San Diego County.....	60
Molybdenite near Bishop, Inyo County.....	61
Molybdenite at the Buchanan prospect, Fresno County.....	62
Molybdenite near Corona, Riverside County.....	62
Molybdenite near California Hot Springs, Tulare County.....	62
Molybdenite at Caliente, Kern County.....	63
Colorado.....	63
Molybdenite mine of Primos Chemical Co., near Empire.....	64
Salamander and Blue Valley molybdenite claims, near Breckenridge, Summit County.....	67
Molybdenite on Bartlett Mountain, Summit County.....	68
Molybdenite near Ophir, San Miguel County.....	69
Other occurrences of molybdenite in Colorado.....	70
Deposit near Robinson, Summit County.....	70
Deposit near Nathrop, Chaffee County.....	70
Prospect near Parkdale, Fremont County.....	70
Specimens from Westcliffe, Custer County.....	71
Claim near Marble, Gunnison County.....	71
Deposit south of Aspen, Pitkin County.....	71
Claims near Kokomo, Summit County.....	72
Property near St. Cloud, Larimer County.....	72
Montana.....	72
Molybdenite near Chico Hot Springs, Park County.....	73
Molybdenite in Carpenter Gulch, Powell County.....	75
Other molybdenite occurrences in Montana.....	76
Wulfenite ore near Twin Bridges, Madison County.....	76

Part II. Description of deposits—Continued.	Page.
New Mexico.....	77
Molybdenite property of the Romero Mining Co.....	78
Washington.....	79
Crown Point mine.....	79
Molybdenite near Loomis, Okanogan County.....	83
Other occurrences of molybdenite in Washington.....	84
Deposit near Safety Harbor Creek, Chelan County.....	84
Deposit on Sheep Mountain, Okanogan County.....	84
Prospects near Oroville, Okanogan County.....	85
Miscellaneous occurrences.....	85
Tabulation of molybdenite and of wulfenite occurrences in the United States.....	86
Part III. Concentration.....	91
Importance of successful concentration of low-grade ores.....	91
Concentration of molybdenite ores.....	92
Rolling and screening processes.....	93
Electrostatic processes.....	94
Principles of separation.....	94
Range of application.....	95
Description of separators.....	96
Concentration tests.....	97
Suggested method for treatment of a typical ore.....	99
Flotation processes.....	101
Water-flotation processes.....	102
Character of water flotation.....	102
Necessary factors in successful water flotation.....	102
Results of tests with Wood's process.....	103
Oil-flotation processes.....	105
Results with Elmore vacuum flotation.....	107
Removal of accompanying metallic sulphides.....	108
Summary.....	109
Concentration of wulfenite ores.....	110
Concentration tests of wulfenite ore from Old Yuma mine, near Tucson, Ariz.....	111
Discussion of results of tests.....	113
Boykin & Hereford wulfenite mill at Mammoth, Ariz.....	115
Discussion of flow sheet.....	116
Results of screen test.....	119
Selected bibliography on molybdenum.....	121
Publications on mineral technology.....	126

---

## ILLUSTRATIONS.

---

	Page.
PLATE I. <i>A</i> , Molybdenite crystal from Crown Point mine, Railroad Creek, Chelan County, Wash.; <i>B</i> , Another molybdenite crystal from Crown Point mine.....	8
II. <i>A</i> , Molybdenite crystal from near Buena Vista, Colo.; <i>B</i> , Flake from molybdenite crystal showing alteration to molybdite, from Santa Maria property, San Diego County, Cal. ....	8

	Page.
PLATE III. <i>A</i> , Molybdenite on quartz from Crown Point mine, Railroad Creek, Chelan County, Wash.; <i>B</i> , Wulfenite crystals lining geode from Mammoth mine, Pinal County, Ariz.....	8
IV. <i>A</i> , Wulfenite crystals from Yuma County, Ariz.; <i>B</i> , Wulfenite crystals from Lucky Bill mine, near Santa Rita, N. Mex., showing unusual forms.....	10
V. <i>A</i> , Wulfenite crystals from Eureka, Nev.; <i>B</i> , Molybdenite crystals on iron-stained granite from Santa Maria, San Diego County, Cal....	10
VI. Map of the vicinity of Mammoth, Ariz.....	46
VII. <i>A</i> , Buildings and dumps of Mammoth and Collins mines, Schultz, Ariz.; <i>B</i> , Crystallized wulfenite in lead carbonate from Mammoth mine, Schultz, Ariz.....	46
VIII. <i>A</i> , Near view of outcrop of Whale vein, Copper Canyon, near Copperville, Ariz.; <i>B</i> , Outcrop of Whale vein, looking south.....	52
IX. <i>A</i> , Section through typical ore from Whale vein, Copper Canyon, near Copperville, Ariz.; <i>B</i> , Section of typical molybdenite ore from Leader mine, Helvetia, Ariz.....	52
X. <i>A</i> , Outcrop of dike on Santa Maria property, San Diego County, Cal., looking northwest; <i>B</i> , Outcrop of low-grade molybdenite ore at Primos mine near Empire, Colo.....	60
XI. <i>A</i> , Molybdenite in iron-stained granite from Santa Maria property, San Diego County, Cal.; <i>B</i> , Molybdenite in granite from John Fletcher quarry, near Corona, Cal.....	60
XII. <i>A</i> , Section through typical molybdenite ore from Primos mine near Empire, Colo.; <i>B</i> , Typical molybdenite ore from Bartlett Mountain, Summit County, Colo.....	64
XIII. <i>A</i> , View of Red Mountain, near Empire, Colo.; <i>B</i> , Section through typical molybdenite ore from Great Western mine, near Chico, Park County, Mont.....	64
XIV. Map of part of Summit County, Colo., showing location of molybdenite deposits on Quandary, Bartlett, and Chalk Mountains....	66
XV. <i>A</i> , Typical molybdenite ore from Salamander claim, Quandary Peak, Summit County, Colo.; <i>B</i> , Scraping tailings for retreatment to recover wulfenite, Mammoth, Ariz.....	66
XVI. <i>A</i> , Section through molybdenite ore from Quigley mine, Carpenter Gulch, near Ophir, Powell County, Mont.; <i>B</i> , Section through molybdenite ore from Romero mine, Porvenir, San Miguel County, N. Mex.....	74
XVII. <i>A</i> , Outcrop of upper vein at mouth of Tunnel No. 1, Crown Point mine, Railroad Creek, Chelan County, Wash.; <i>B</i> , Lower tunnel and dump from upper workings, Starr mine, near Loomis, Okanogan County, Wash.....	80
XVIII. <i>A</i> , Tailing piles containing wulfenite from stamp and cyanide mills, Mammoth, Ariz.; <i>B</i> , Boykin and Hereford wulfenite mill, Mammoth, Ariz.....	114
FIGURE 1. Sketch map of the vicinity of Lake Chelan, Wash., and vicinity, showing the situation of the Crown Point and Robischand mines.	80
2. Flow sheet of Boykin & Hereford wulfenite mill, Mammoth, Ariz.	117



# MOLYBDENUM; ITS ORES AND THEIR CONCENTRATION.

---

By FREDERICK W. HORTON.

---

## INTRODUCTION.

With large deposits of low-grade molybdenum ore and a latent market for molybdenum, which with development might perhaps equal that for tungsten, the United States has for years made no production of molybdenum ore worthy of mention. In fact, during the decade previous to 1914, there was no output of molybdenum ore in this country except in 1905, 1906, and 1907, when a few small lots of molybdenite and wulfenite were marketed.<sup>a</sup>

To ascertain why this possible source of economic wealth should remain undeveloped, the United States Bureau of Mines undertook an investigation in 1914, with a view of assisting the establishment of a molybdenum industry in this country. A preliminary review of the situation showed the following conditions:

Briefly there was a lack of any considerable demand for molybdenum for the one known use that might consume any large tonnage of it, that is, for the manufacture to certain alloy steels to which molybdenum imparts properties roughly similar to those produced by tungsten. It was at once evident that the market for molybdenum in alloy steels must be developed, or that extensive new uses for the metal must be created, before the demand would be sufficient to warrant any extensive mining of molybdenum ore.

Investigation showed that one of the chief factors in retarding the development of any demand for molybdenum by the alloy-steel trade was that manufacturers who might use, or might investigate the possibilities of using, the metal were kept out of the market by the fear of not being able to obtain steady supplies. On the other hand, those who might be interested in the development of some of the extensive low-grade molybdenite or wulfenite deposits in this country were prevented from doing so by the small visible demand and the fear that any large production would glut the market. Owing to these conditions the mining of molybdenum has in the past been confined

---

<sup>a</sup> Mineral Resources U. S. for 1904 to 1913, U. S. Geol. Survey, 1905 to 1914.

almost entirely to small-scale operations on high-grade streaks of molybdenite ore, and the methods of recovery have been limited largely to cobbing and hand picking.

#### SCOPE AND PURPOSE OF INVESTIGATION.

The problem before the Bureau of Mines was to ascertain the character and extent of the deposits of molybdenum ore in the United States from which supplies requisite for the development of the market might be obtained, and how the ores might best be concentrated into a marketable product. These features were the ones that the investigation described in the following pages principally covered.

The direct purpose of this bulletin is, on the one hand, to prove to possible consumers of molybdenum that the element is not as rare as commonly supposed, and that this country possesses many deposits of low-grade ore from which large supplies may be derived, and on the other hand, to prove to present and prospective producers of molybdenum that there is a latent market for their product in the alloy-steel trade, which needs only the assurance of steady supplies for a considerable development. It is also intended to assist the miner of molybdenum by giving him information regarding various methods of concentration that have proven applicable to the different types of molybdenum ore, and by acquainting him with market conditions, uses, prices, etc. The various molybdenum minerals are described in detail, and tests for their determination are given to aid the prospector in recognizing them. In addition to the most applicable and reliable qualitative tests for molybdenum, the quantitative determination of the element is described in some detail, as the methods of analyses recommended in the ordinary textbooks will not in general give correct results, particularly on low-grade ores.

#### PRESENT CONDITION OF THE MOLYBDENUM INDUSTRY.

The unprecedented demand for steel-hardening metals occasioned by the European war is slowly drawing the attention both of prospective consumers and of producers to possibilities in regard to molybdenum. Several mining operations looking toward a considerable production of both molybdenite and wulfenite have already been commenced, and manufacturers are investigating the possibilities of using molybdenum in a large way. The fact that molybdenum is now (April 1, 1916) selling at \$3 to \$4 a pound, as compared with \$8 to \$10 a pound for tungsten, and the further fact that 1 pound of molybdenum will produce approximately the same results as 2 or 3 pounds of tungsten, are serving to quicken this interest.

Under the influence of the high prices being paid for the metal and the unusual demand, the whole molybdenum industry is developing

as never before, not only in the United States, but also in Queensland, New South Wales, and Norway, the three countries that have previously supplied practically the entire world's production. In this country, since late in 1914, a fair-sized molybdenite mine has been opened up on a low-grade deposit at Red Mountain, near Empire, Colo., and extensive shipments of ore have been made. Many molybdenite mines and prospects throughout the West have produced small lots of high-grade ore, and some of them have accumulated dumps of milling ore. A noteworthy amount of development work has been done at many of these properties, and several of them are rapidly being put into condition to become producers. Since the summer of 1914 a considerable tonnage of wulfenite concentrates of medium grade has been derived by milling operations on some large tailing piles at Mammoth, Ariz., and in the spring of 1915 the output from this source was more than doubled. It is lately reported that a mill has been installed to treat wulfenite ore from the Old Yuma mine, near Tucson, Ariz.

Developments have already progressed to such an extent that, in the author's opinion, the molybdenum production of the United States in 1915 will compare favorably with that of Queensland, New South Wales, or of Norway, and it may even exceed them.

#### THE FUTURE OF THE MOLYBDENUM INDUSTRY.

Of course, the extremely high prices that are now being paid for molybdenum can not last, but even under normal conditions the author sees no reason why molybdenum should not compete with tungsten in the manufacture of many alloy steels. It is surely as plentiful an element as tungsten, its ores can be mined as cheaply as those of the latter metal, and although high-grade concentrates from molybdenite ores will probably cost a little more to produce than from tungsten ores, the fact that much less molybdenum need be used to produce a given result should give it the advantage in the market. If steady supplies of molybdenum can be obtained at or near the prices at which tungsten is offered, the demand for it can not help but develop greatly within a few years.

#### ACKNOWLEDGMENTS.

The writer takes pleasure in acknowledging his indebtedness to Mr. Karl L. Kithil, mineral technologist of the Bureau of Mines, for his kindly advice in the preparation of this report, and to Dr. Charles L. Parsons, chief of the division of mineral technology of the bureau, for many valuable suggestions. The author's thanks are also extended to Messrs. R. B. Moore, J. C. Morgan, and H. A. Doerner, of the Denver office of the bureau, for chemical analyses, and to Dr.

F. B. Laney, of the United States Geological Survey, for photographing ore specimens and for petrographic work; also to R. E. Head, of the Salt Lake office of the bureau, for the preparation of rock sections and for assistance in concentration tests:

The writer expresses his appreciation to the following companies and individuals for their kindness in placing at his disposal the equipment of their testing laboratories for concentration work: W. G. Swart, of the American Zinc Ore Separating Co.; Henry E. Wood, president of Henry E. Wood & Co.; Albert M. Plumb, of the Plumb Jig Co.; Frank E. Shepard, president of the Denver Engineering Works Co.; and W. S. Myers, president of the Sutton, Steele & Steele Manufacturing, Mining & Milling Co., all of Denver, Colo. Acknowledgments are also made to the following individuals and companies for their courtesy in supplying the writer with molybdenum ores for concentration tests: George H. Daily, Tucson, Ariz.; R. J. A. Widmar, Breckenridge, Colo.; William Young and C. S. Lively, Empire, Colo.; H. Leal, Denver, Colo.; Smith & Sawyer, Kingman, Ariz.; Young Bros., Shultz, Ariz.; Santa Maria Molybdenum Mining & Milling Co., San Diego, Cal.; and the Grand View Mining & Development Co., Twin Bridges, Mont.

The writer's thanks are also due E. S. Dana and William E. Ford, of Yale University; Frank L. Hess, of the United States Geological Survey; John E. Wolff, of Harvard University; Charles H. Warren, of the Massachusetts Institute of Technology; George D. Merrill, of the United States National Museum; L. P. Gratacap, of the American Museum of Natural History; and Oliver C. Farrington, of the Field Museum of Natural History, for information as to occurrences of molybdenum minerals; E. R. Boericke and A. G. Pohnsdorf, of Denver, Colo., for photographs; the Colorado Bureau of Mines and A. B. Frenzel, of Denver, Colo., for the loan of specimens; and all those companies and individuals who have allowed him to examine their molybdenum properties or have otherwise aided him.



## PART I. THE MOLYBDENUM MINERALS.

The following table lists the known molybdenum minerals and gives their theoretical composition, and the percentage of molybdenum each mineral contains as based on this composition:

### *Molybdenum minerals.*

Mineral.	Composition.	Per cent molybdenum.	Mineral.	Composition.	Per cent molybdenum.
Molybdenite <i>a.</i>	MoS <sub>2</sub> .....	59.95	Powellite.....	CaMoO <sub>4</sub> .....	47.98
Wulfenite.....	PbMoO <sub>4</sub> .....	26.15	Pateraite.....	CoMoO <sub>4</sub> .....	43.84
Molybdite.....	Fe <sub>2</sub> O <sub>3</sub> 3MoO <sub>3</sub> $\frac{7}{4}$ H <sub>2</sub> O.....	39.63	Achrematite....	3(3Pb <sub>3</sub> As <sub>2</sub> O <sub>8</sub> PbCl <sub>2</sub> ) 4= (Pb <sub>2</sub> MoO <sub>6</sub> ).	3.40
Ilsemanite <i>b.</i>	MoO <sub>3</sub> 4MoO <sub>3</sub> ( $\frac{7}{4}$ ).....	68.18	Eosite.....	Vanado-molybdate of lead.	.....
Belonesite.....	MgMoO <sub>4</sub> .....	52.08			

*a* An amorphous variety of molybdenite called jordisite is said to exist (see p. 9.)

*b* Composition doubtful (see p. 12).

Of the above list only the first two minerals, molybdenite and wulfenite, have thus far proved of commercial importance. With the exception of molybdite, the other minerals listed are rare and are to be considered only in the light of interesting mineralogical occurrences. Besides these there are other molybdenum minerals that have been reported, but information concerning them is so incomplete both as regards occurrence and composition that their existence as definite species can not be considered as authenticated. (See p. 12.)

### MOLYBDENITE.

Molybdenite is the disulphide of molybdenum (MoS<sub>2</sub>) and contains 59.95 per cent of molybdenum and 40.05 per cent of sulphur. It is a soft, opaque, lead-gray mineral with a metallic luster and greasy feel. It commonly occurs in flakes or scales having a prominent basal cleavage and resembling some micas in the way the flakes may be split into thin leaves. Finely granular and massive forms are also common. The mineral is sectile and in the flaky form the laminæ are flexible but not elastic. In hardness it ranges from 1 to 1.5, being so soft that it soils the fingers readily in handling and marks paper, on which it leaves a bluish-gray trace. On porcelain its streak is slightly greenish. Its specific gravity has been variously determined at 4.7 to 4.8.<sup>c</sup>

<sup>c</sup> Dana, E. S., A system of mineralogy, 6th ed., 1911, p. 41.

On account of many similar characteristics molybdenite is often confused with graphite, but it may be easily distinguished from the latter as graphite has a much lower specific gravity, 2.09 to 2.23, and its streak is lead-gray on both paper and porcelain. Heating a fragment of the mineral in a closed tube will conclusively settle any further question as to its identity, as the strong sulphurous odor given off by molybdenite is entirely lacking with graphite.

Molybdenite crystallizes in hexagonal form and the crystals are tabular or short and slightly tapering prisms. The prismatic planes are horizontally striated and on the base of some crystals there are striae normal to the edges. In Plate I, which illustrates characteristic forms of molybdenite crystals, the horizontal striæ on the prismatic faces may readily be seen. Many molybdenite aggregates exhibit a distinct radial formation. This typical structure is well shown in Plate II, A. A typical molybdenite ore is shown in Plate III, A.

An artificial crystallized molybdenite with a specific gravity of 5.06 has been made by melting together potassium carbonate, sulphur, and molybdic oxide in a platinum crucible.<sup>a</sup>

Molybdenite commonly alters to molybdite, but it is also reported as sometimes altering to ilsemanite. (See pp. 12 and 13.)

#### TESTS FOR MOLYBDENITE.

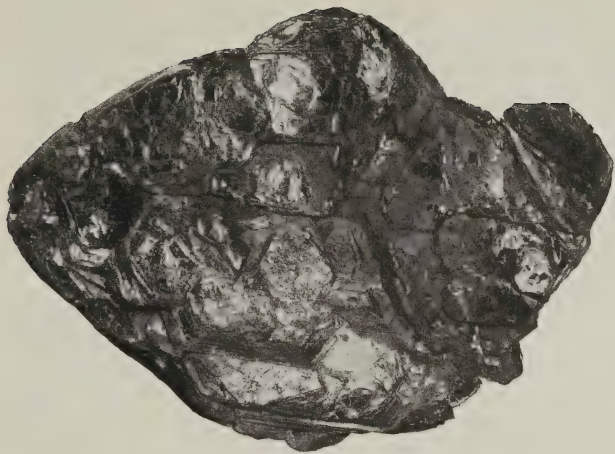
Different methods of testing molybdenite follow:<sup>b</sup> When heated in an open tube molybdenite gives off sulphurous fumes, and a pale yellow crystalline sublimate of molybdenum trioxide ( $\text{MoO}_3$ ) is formed. Before the blowpipe, the mineral is infusible and imparts a yellowish-green color to the flame. On charcoal in the oxidizing flame, pulverized molybdenite gives a strong odor of sulphur and the charcoal is coated with crystals of molybdic oxide which appears yellow while hot and white when cold. Near the heated mineral the coating is copper-red, and if the white coating is touched with an intermittent reducing flame, it becomes a beautiful azure blue. Molybdenite is decomposed by nitric acid, leaving a white or grayish residue (molybdic oxide).

#### OCCURRENCES OF MOLYBDENITE.

Probably three-fourths of the reported occurrences of molybdenite are in acid igneous rocks such as granites, pegmatites, trachytes, and syenites. Next in order of importance as regards deposits of the mineral are the metamorphic rocks, molybdenite being found in serpentines and gneisses, and in amphibolite, chlorite, talc, and mica schists, etc. Next follow the sedimentary rocks with occur-

<sup>a</sup> Dana, E. S., op. cit., p. 42.

<sup>b</sup> Dana, E. S., op. cit., p. 41.



A. MOLYBDENITE CRYSTAL FROM CROWN POINT MINE, RAILROAD CREEK,  
CHELAN COUNTY, WASH. (TWO-THIRDS NATURAL SIZE.)

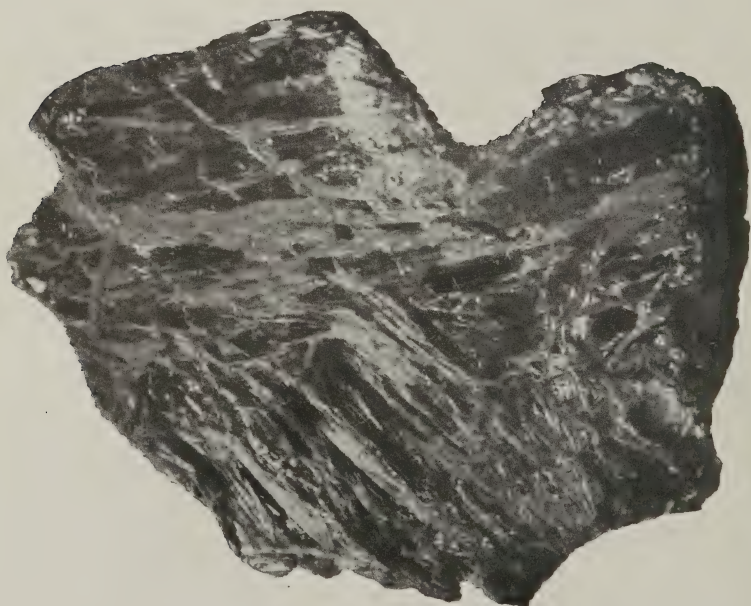


B. ANOTHER MOLYBDENITE CRYSTAL FROM CROWN POINT MINE.  
(ONE AND ONE-HALF TIMES NATURAL SIZE.)



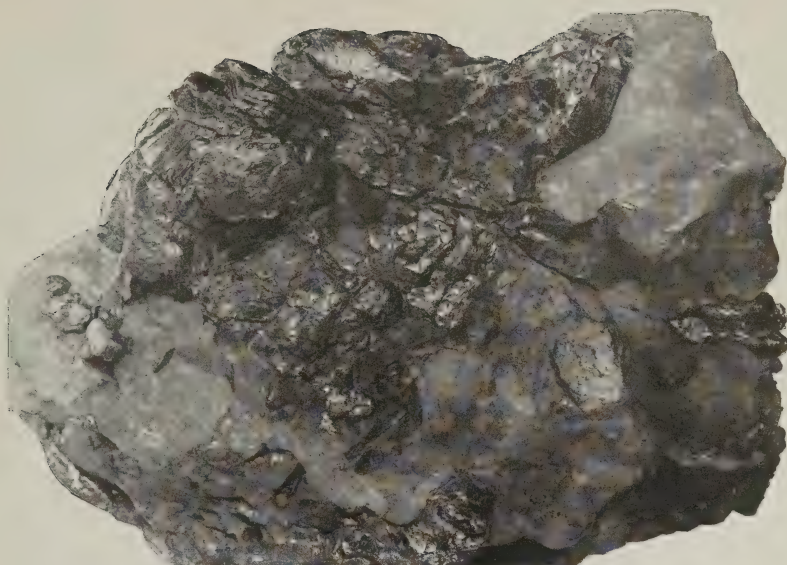


A. MOLYBDENITE CRYSTAL FROM NEAR BUENA VISTA, COLO. (ONE AND ONE-HALF TIMES NATURAL SIZE.)



B. FLAKE FROM MOLYBDENITE CRYSTAL SHOWING ALTERATION TO MOLYBDITE, FROM SANTA MARIA PROPERTY, SAN DIEGO COUNTY, CAL. (FOUR TIMES NATURAL SIZE.)





A. MOLYBDENITE ON QUARTZ FROM CROWN POINT MINE, RAILROAD CREEK, CHELAN COUNTY, WASH. (THREE-FIFTHS NATURAL SIZE.)



B. WULFENITE CRYSTALS LINING GEODE FROM MAMMOTH MINE, PINAL COUNTY, ARIZ. (ONE-HALF NATURAL SIZE.)

LIBRARY  
OF THE  
UNIVERSITY OF ILLINOIS

rences of the mineral in conglomerates and limestones, and finally the basic igneous rocks with a few deposits in gabbros, basalts, etc.

Localities in the United States where molybdenite has been reported are named in part 2 of this report, and detailed descriptions of many characteristic occurrences are given there.

#### ACCOMPANYING MINERALS.

Besides the occurrence of gold and of various silver minerals with molybdenite, the author has noted the following accompanying minerals: Molybdite, bornite, chalcopyrite, malachite, azurite, chrysocolla, native copper, pyrite, galena, pyrrhotite, magnetite, limonite, sphalerite, bismuthinite, cassiterite, wolframite, scheelite, calcite, quartz, orthoclase, pyroxene, hornblende, garnet, epidote, tourmaline, muscovite, and biotite. The following additional minerals have been recorded by Crook<sup>a</sup> as associated with molybdenite: Arsenopyrite, fluorite, barite, apatite, rutile, oligoclase, tremolite, scapolite, and zircon; and Umpleby<sup>b</sup> has noted tetrahedrite, chalcocite, cuprite, hubnerite, and manganese oxides as accompanying minerals. Dolomite, talc, beryl, phlogophite, and löllingite are also mentioned by Hintze<sup>c</sup> as occurring with molybdenite.

#### JORDISITE.

According to Cornu,<sup>d</sup> there exists a black powdery form of molybdenum sulphide that he considers to be distinct from crystalline molybdenite. It is described as "colloidal molybdenum sulphide," and is said to alter to ilsemanite<sup>e</sup> and to occur at Himmelfurst, Freiberg, Saxony. Jordisite is listed as a variety of molybdenite by Cahen and Wootton.<sup>e</sup>

#### WULFENITE.

Wulfenite is a molybdate of lead ( $\text{PbMoO}_4$ ) and theoretically contains 26.15 per cent of molybdenum and 56.42 per cent of lead. It is a heavy, brittle, subtransparent to subtranslucent mineral with a resinous or adamantine luster and is generally of a wax or orange-yellow color. It may, however, be siskin and olive green, yellowish-gray, brown, grayish-white to nearly colorless, or orange to bright red. Its hardness is 2.75 to 3 and its specific gravity is 6.7 to 7. It has a subconchoidal fracture and a white streak. In planes parallel with its crystal pyramid faces it has a smooth cleavage, but in other

<sup>a</sup> Crook, A. R., Bull. Geol. Soc. America, vol. 15, 1904, pp. 283-288.

<sup>b</sup> Umpleby, J. B., Geology and ore deposits of Lemhi County, Idaho: U. S. Geol. Survey Bull. 528, 1913, p. 73.

<sup>c</sup> Hintze, Carl, Handbuch der Mineralogie, Bd. 1, pp. 410-418.

<sup>d</sup> Cornu, F., Natürliches kolloides Molybdänsulfid (Jordisite): Ztschr. Chem. Ind. Kolloide, Bd. 4, 1909, p. 190.

<sup>e</sup> Cahen, Edward, and Wootton, W. O., The mineralogy of the rarer metals; a handbook for prospectors. 1912, pp. 51-52.

directions the cleavage is less distinct. It crystallizes in the tetragonal system with pyramidal hemihedrimism. The crystals are commonly square and tabular and are sometimes extremely thin, with a vicinal pyramid replacing the basal plane. Less frequently the crystals are octahedral or prismatic, the prismatic faces showing the hemihedrimism characteristic of the mineral. Plates III, *B*, and IV, *A*, illustrate common tabular forms of wulfenite crystals and Plate V, *A*, shows a fine specimen illustrating the extremely thin crystals referred to above. Wulfenite crystals of mineral form are illustrated in Plate IV, *B*.

Wulfenite generally occurs in well-crystallized forms, but it is also found in coarse-grained or fine-grained masses. Small percentages of calcium, chromium, vanadium, copper, iron, or aluminum are generally present in wulfenite as impurities.

Among others, the following characteristic analyses of American wulfenites are given by Dana:<sup>a</sup>

*Characteristic analyses of American wulfenites.*

Source of sample.	Per cent MoO <sub>3</sub> .	Per cent PbO.	Per cent other oxides.	Total per cent.
Eureka County, Nev.....	39.33	61.11	1.04 CaO <sub>3</sub> , 0.38 Fe <sub>2</sub> O <sub>3</sub>	101.86
Phoenixville, Pa.....	39.21	60.00	.38 Cr <sub>2</sub> O <sub>3</sub>	99.59
Do.....	37.47	60.30	1.28 V <sub>2</sub> O <sub>5</sub>	99.05

**TESTS.<sup>b</sup>**

Before the blowpipe wulfenite decrepitates and fuses below 2. With borax in the oxidizing flame it gives a colorless glass which in the reducing flame becomes an opaque black or dirty green with black flocks. With salt of phosphorus in the oxidizing flame wulfenite gives a yellowish-green glass which becomes dark green in the reducing flame. The powdered mineral, when heated with soda on charcoal, yields metallic lead. On evaporation with hydrochloric acid wulfenite is decomposed, forming lead chloride and molybdic oxide. Moistening the residue with water and adding metallic zinc gives an intense blue color which persists after dilution.

**OCCURRENCE AND ACCOMPANYING MINERALS.**

Deposits of wulfenite are confined almost wholly to veins, in which it occurs associated with other lead minerals. Localities in the United States where wulfenite has been reported are named in part 2, and detailed descriptions of several characteristic occurrences are also given.

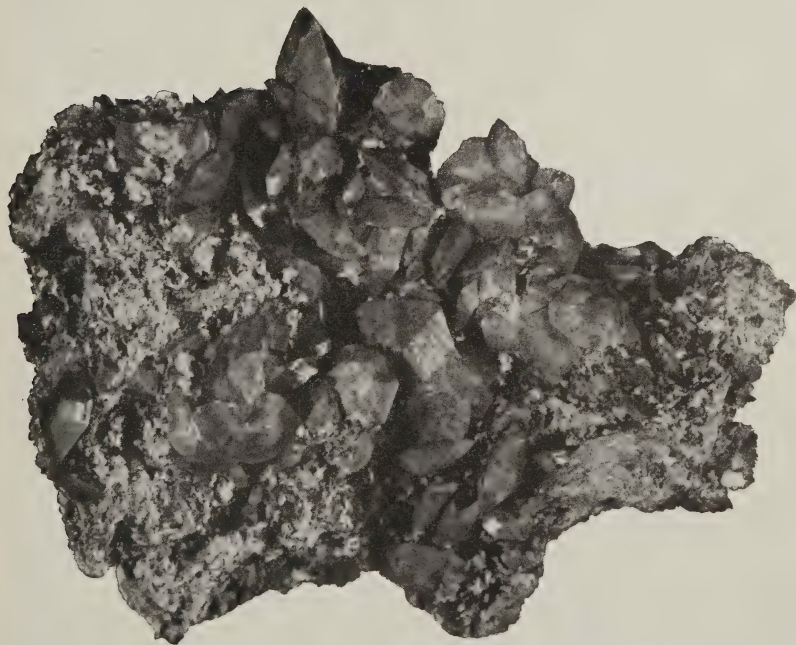
<sup>a</sup> Dana, E. S., A system of mineralogy, 6th ed., 1911, p. 991.

<sup>b</sup> Dana, E. S., loc. cit.

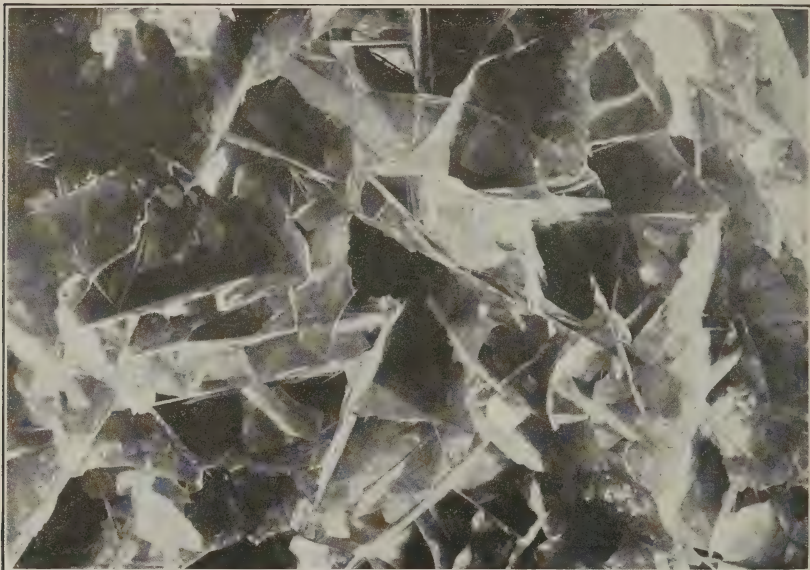




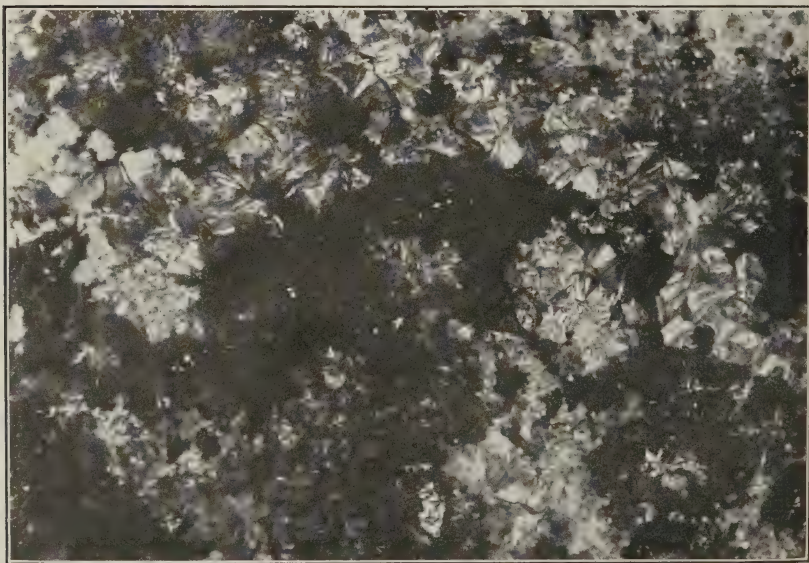
A. WULFENITE CRYSTALS FROM YUMA COUNTY, ARIZ. (NATURAL SIZE.)



B. WULFENITE CRYSTALS FROM LUCKY BILL MINE, NEAR SANTA RITA, N. MEX., SHOWING UNUSUAL FORMS. (ONE AND ONE-FOURTH TIMES NATURAL SIZE.)



A. WULFENITE CRYSTALS FROM EUREKA, NEV. (NATURAL SIZE.)



B. MOLYBDITE CRYSTALS ON IRON-STAINED GRANITE FROM SANTA MARIA, SAN DIEGO COUNTY, CAL. (THREE TIMES NATURAL SIZE.)

Outside of the gangue-forming minerals, wulfenite has been found associated with gold and various silver minerals, with galena, cerussite, pyromorphite, vanadinite, anglesite, and descloizite. In connection with the occurrence of gold with wulfenite, it may be of interest to note that the native gold is sometimes contained directly in the wulfenite crystals themselves. (See p. 115.)

### MOLYBDITE.

Molybdate is a hydrous ferric molybdate ( $\text{Fe}_2\text{O}_3 \cdot 3\text{MoO}_3 \cdot 7\frac{1}{2}\text{H}_2\text{O}$ ) and theoretically contains 39.63 per cent of molybdenum. It is a lemon-yellow to pale-yellow mineral, and occurs as an earthy powder, incrustations, fibrous masses, or capillary crystals in radiating groups. The crystals and many of the fibrous forms of the mineral have a silky luster. Molybdate is an alteration product of molybdenite, probably formed by the interaction of molybdic acid (liberated by the oxidation of the molybdenite) and limonite. The formation of the limonite in many cases may have resulted from the weathering of pyrite, which is commonly associated with the molybdenite, simultaneously with or previous to the weathering of the molybdenite.

Plate V, *B*, illustrates a group of molybdate crystals, showing their capillary form and characteristic grouping in radiating clusters. Plate II, *B*, shows a flake from a large crystal of molybdenite, with molybdate extensively developed along its cleavage planes.

For many years the erroneous idea has existed that molybdate agrees in composition with the artificial molybdic trioxide obtained by oxidizing molybdenite. The present textbooks on mineralogy almost universally persist in this error, giving the composition of molybdate as  $\text{MoO}_3$ , and stating that the pure mineral contains 66.7 per cent of molybdenum. That such is not the case was first called to the author's attention by chemical tests of a sample of molybdate which from microscopic examination seemed to be practically pure. The presence of a large percentage of iron was determined, and the same result obtained from tests on other samples of molybdate from different localities led the author to believe that the iron was an essential part of the mineral. A search of the literature on molybdate revealed the fact that Schaller<sup>a</sup> had not only proved the presence of combined iron in molybdate, but had definitely established its composition as  $\text{Fe}_2\text{O}_3 \cdot 3\text{MoO}_3 \cdot 7\frac{1}{2}\text{H}_2\text{O}$ .

His analyses of molybdate from different localities in the United States, given in the accompanying table, have the insoluble matter deducted and are recalculated to 100 per cent. The last column of

<sup>a</sup> Schaller, W. T., Mineralogical notes—series 1: U. S. Geol. Survey Bull. 490, 1911, pp. 84-92.



the table gives the theoretical analysis as calculated from the formula  $\text{Fe}_2\text{O}_3 \cdot 3\text{MoO}_3 \cdot 7\frac{1}{2}\text{H}_2\text{O}$ . Attention is called to the close agreement of the analysis of the molybdate from New Hampshire with this calculated analysis.

*Analyses of molybdate.*

Constituent.	Telluride, Colo.	Hortense, Colo.	Californ- ia.	New Hamp- shire.	Calcu- lated from formula.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
$\text{H}_2\text{O}$ .....	16.8	20.19	19.5	18.28	18.57
$\text{Fe}_2\text{O}_3$ .....	20.2	20.30	20.0	21.87	22.01
$\text{MoO}_3$ .....	63.0	59.42	60.5	59.85	59.42
	100.00	100.00	100.0	100.00	100.00

The specific gravity of the molybdate from Hortense, Colo., is given by Schaller as 2.99.

### TESTS FOR MOLYBDITE.

Regarding tests of molybdate, Schaller <sup>a</sup> writes as follows:

On heating the mineral in a closed tube abundant water is easily given off and the mineral assumes a dark-olive color; on further heating it again becomes lighter in color. On heating the mineral in a crucible the color changes are very marked. At first the yellow mineral darkens and becomes a dark gray, appearing almost black and with slight olive tint, then it becomes a light yellow again, and on further heating changes to a deep orange color. If the mineral now be allowed to cool, the orange changes to yellow and back to orange again on reheating. If the dark-colored material be allowed to cool, it retains its dark gray color, and on reheating passes through yellow to the orange. On heating for some time at a higher temperature, the mineral, on cooling, becomes a permanent bright green. By further heating all of the molybdenum is volatilized and the dark-red ferric oxide remains. The mineral is readily soluble in hydrochloric acid, and is decomposed by ammonia, taking on a brown color (probably due to the separating ferric hydroxide). After a while, all the molybdenum of the mineral goes into solution, leaving the insoluble ferric hydroxide.

### OCCURRENCE AND ACCOMPANYING MINERALS.

As molybdate is an alteration product of molybdenite, its occurrences and associated minerals are practically the same as those of the latter mineral. (See p. 11.)

### THE RARE MOLYBDENUM MINERALS.

#### ILSEMANNITE.

Ilsemannite <sup>b</sup> is an oxide of molybdenum said to result from the decomposition of metallic molybdates and is probably identical with the artificial compound  $\text{MoO}_2 \cdot 4\text{MoO}_3$ . It is a blue-black to black cryptocrystalline mineral, and has been reported as occurring in

<sup>a</sup> Schaller, W. T., op. cit., pp. 85-86.

<sup>b</sup> Dana, E. S., A system of mineralogy, 6th ed., 1911, p. 202.



barite and associated with wulfenite at Bleiberg, in Corinthia. It is soluble in water, giving a deep blue solution which on evaporation yields dark-blue crystals. Lindgren and Ransome<sup>a</sup> state that material from a quartz vein carrying molybdenite, pyrite, and sphalerite and situated a short distance east of the Howard flat vein in the Cripple Creek District, Colo., upon exposure to the sun and air, turns yellow or green and finally a deep Prussian blue. Little tufts of yellow material are formed on the specimens and occasionally coatings of a dark-blue mamillary substance. "The yellowish material is believed to be molybdite." "The blue substance, which dissolves in water and is associated with some ferrous sulphate, also contains a large amount of molybdenum, and probably is the rare mineral ilsemanite ( $\text{MoO}_2\cdot 4\text{MoO}_3$ )."

#### BELONESITE.

Belonesite<sup>b</sup> is regarded as magnesium molybdate ( $\text{MgMoO}_4$ ). It is a white transparent mineral occurring in minute acicular crystals of the tetragonal system, and theoretically contains 52.08 per cent of molybdenum. It occurs in rock fragments enveloped in the Vesuvian lava of 1872.

#### TESTS.

Before the blowpipe belonesite fuses with difficulty. It dissolves readily in salt of phosphorus, but less easily in the borax bead, and is insoluble in acids.

#### POWELLITE.

Powellite is calcium molybdate ( $\text{CaMoO}_4$ ), which when pure contains 47.98 per cent of molybdenum. It is a dull-gray color, and according to Schaller<sup>c</sup> has a specific gravity of approximately 4.25. It is a pseudomorph after molybdenite, often preserving the structure of the latter mineral and occurring in platy masses. Dana<sup>d</sup> states that calcium tungstate is present in the mineral and quotes an analysis showing 10.28 per cent  $\text{WO}_3$ . Schaller, however, calls attention to the fact that any calcium tungstate present exists as a mechanical mixture of scheelite with the mineral and states that "The association of molybdenite with scheelite is well known, and when the molybdenite alters to powellite the association of powellite with scheelite necessarily follows. The agencies affecting the change from molybdenum sulphide to calcium molybdate are apparently without effect on the scheelite."

<sup>a</sup> Lindgren, Waldemar, and Ransome, F. L., Geology and gold deposits of the Cripple Creek district, Colo.: U. S. Geol. Survey, Prof. Paper No. 54, 1906, p. 124.

<sup>b</sup> Dana, E. S., op. cit., p. 992.

<sup>c</sup> Schaller, W. T., Mineralogical notes—series 1: U. S. Geol. Survey, Bull. 490, 1911, pp. 80-83.

<sup>d</sup> Dana, E. S., op. cit., p. 989.

## TESTS.

The mineral fuses at about 5 to a gray mass. It reacts for molybdenum with salt of phosphorus and is decomposed by nitric and hydrochloric acids.<sup>a</sup>

## OCCURRENCE.

Powellite has been found in the United States in the Peacock lode of the Seven Devils mining district in western Idaho, where it is associated with argentiferous bornite and dark-brown garnet;<sup>a</sup> at the South Hecla copper mine, Houghton County, Mich., where it occurs in fine crystals;<sup>b</sup> near Oak Springs, Nye County, Nev.;<sup>c</sup> and Baringer Hill, Llano County, Tex.<sup>d</sup>

## PATERAITE.

Pateraite<sup>e</sup> is supposed to be cobalt molybdate ( $\text{CoMoO}_4$ ). It occurs as an impure massive mineral of black color associated with uranium in the Elias mine, Joachimisthal, Austria.

## ACHREMATITE.

Achrematite<sup>f</sup> is a massive, crypto-crystalline mineral with the formula  $3(\text{Pb}_3\text{As}_2\text{O}_8\text{PbCl}_2)4(\text{Pb}_2\text{MoO}_5)$  and contains 3.40 per cent of molybdenum. It has an uneven to subconchoidal fracture, is brittle, has a hardness of 3 to 4, and a specific gravity of about 6. In color it is sulphur-yellow to orange and red, but in mass is liver-brown, owing to admixed limonite. Its streak is pale cinnamon-brown; its luster is resinous to adamantine and fragments of the mineral are translucent on thin edges. It occurs at the mines of Guanacere, Chihuahua, Mexico.

## TESTS.

Before the blowpipe it decrepitates slightly, turning a dark brick red and fusing easily to a nearly black globule which shows indistinct crystalline facets on cooling. On charcoal it yields arsenical odors, a lead coating, and finally globules of lead

## EOSITE.

Eosite<sup>g</sup> is supposed to be a vanado-molybdate of lead. Its exact composition is not considered as definitely established, although Gin<sup>h</sup>

<sup>a</sup> Dana, E. S., A system of mineralogy, 6th ed., 1911, p. 989.

<sup>b</sup> Dana, E. S., op. cit., appendix I, p. 55.

<sup>c</sup> Schaller, W. T., Mineralogical notes—series 1: U. S. Geol. Survey, Bull. 490, 1911, pp. 80-83.

<sup>d</sup> Schaller, W. T., loc. cit.

<sup>e</sup> Dana, E. S., op. cit., pp. 991-992.

<sup>f</sup> Dana, E. S., op. cit., p. 992.

<sup>g</sup> Dana, E. S., loc. cit.

<sup>h</sup> Gin, G., A memoir on the methods of treatment of simple and complex ores of molybdenum, tungsten, uranium, and vanadium: Trans. Am. Elec. Chem. Soc., vol. 12, 1907, p. 412.

gives its formula as  $\text{Pb}_3\text{V}_2\text{MoO}_{16}$ . It occurs in minute square octahedral crystals of the tetragonal system which are found implanted on pyromorphite and cerussite at Leadhills, Scotland. It has a hardness of 3 to 4, and a brownish, orange-yellow streak. In color it is deep aurora-red.

TESTS.<sup>a</sup>

When heated in a closed tube, eosite darkens, but regains its color on cooling. When fused with potassium bisulphate it gives a mass that is slightly yellow while hot, but on cooling turns first reddish-brown, and finally a brownish orange-yellow. This mass, dissolved in water and boiled with metallic tin, colors the solution a faint greenish-blue. Eosite is not so rapidly attacked by hydrochloric acid as wulfenite or crocoite. When a splinter of the mineral is placed on a glass plate and treated with hydrochloric acid, with the subsequent addition of alcohol, and then gently evaporated, a blue to bluish-green coating is formed with a green precipitate on the edges.

## A NEW MOLYBDENUM MINERAL.

F. B. Laney, of the United States Geological Survey, recently discovered a dark-green mineral rich in molybdenum, which occurs as an incrustation on specimens of ore from Vein No. 21 of the Lucania Tunnel, near Idaho Springs, Colo. Dr. Laney's examination of the occurrence, made in conjunction with the author, showed that the mine water, which percolated slowly from the vein in the immediate neighborhood of the green incrustations, contained a molybdenum compound that colored it a deep blue. This color was so intense that in several experiments in which the mine water was used as a writing fluid the writing was as legible as with an ordinary ink. The acidity of the water was pronounced, and in some places where it had dripped upon the iron mine rails it had eaten them to a depth of three-quarters of an inch or more.

An analysis of a sample of this water by R. C. Wells, of the United States Geological Survey, showed the following results:

*Analysis of mine water from Vein 21, Lucania Tunnel, Idaho Springs, Colo.<sup>b</sup>*

Substance.	Grams per liter.	Substance.	Grams per liter.
Molybdenum trioxide ( $\text{MoO}_3$ ).....	7.98	Magnesium.....	0.73
Molybdenum dioxide ( $\text{MoO}_2$ ).....	Trace.	Sodium.....	.26
Ferrous iron.....	2.01	Potassium.....	.14
Ferric iron.....	1.75	Sulphate ( $\text{SO}_4$ ).....	18.26
Aluminum.....	.27	Chlorine (Cl).....	.17
Calcium.....	.50	Hydrogen (of free acid).....	.13

<sup>a</sup> Dana, E. S., loc. cit.

<sup>b</sup> The specific gravity of this water at 25° C. was 1.031.

There is no doubt that the color of the water is due to a dissolved molybdenum blue, possibly ilsemanite, but just what the compound is has not been determined.

The blue mine water had impregnated considerable areas of the vein material in the drift on both sides of the main tunnel and immediately adjoining it. The rock was colored a light blue with the coloring particularly strong along the fractures in the vein material and in the more porous rock. Where the rock carrying the molybdenum blue had been exposed to the air circulating along the main tunnel and about the mouths of the drifts there was a powdery incrustation of the dark-green molybdenum mineral already referred to. Several specimens were collected where this incrustation was a yellowish or golden green and a few where it was almost a bright yellow. On the dump at the mouth of the main tunnel many specimens were coated with the green molybdenum mineral. The latter had seemingly been formed by the action of the air on ore that had been impregnated with the molybdenum blue.

Owing to the difficulty of obtaining reasonably pure samples of these blue and green minerals, and to the inherent difficulties in the analysis of compounds, whose state of oxidation is so unstable, their exact composition has not as yet been determined.

The molybdenum minerals described above are probably identical with those noted by Lindgren and Ransome <sup>a</sup> in a small vein east of the Howard flat vein in the Cripple Creek district, Colorado. However, the author believes that the green molybdenum mineral (relatively insoluble) is formed from the soluble blue compound, and not vice versa, as indicated by Lindgren and Ransome.

#### DOUBTFUL SPECIES.

As already stated, there are other molybdenum minerals that have been reported besides those described above. Definite information concerning them is, however, so meager that their existence as distinct species is problematical. Principal among these doubtful minerals are molybdurane, molybdoferrite and knightite. The first two are referred to by Moissan <sup>b</sup> and also by Gin <sup>c</sup> who give the formula of molybdurane as  $\text{UO}_2 \cdot \text{UO}_3 \cdot 2\text{MoO}_4$ , and of molybdoferrite as  $\text{FeMoO}_4$ . Des Cloizeaux <sup>d</sup> speaks of molybdurane as a molybdate of uranium, and says that it is found at Joachimsthal, Bohemia. Knightite, said to be a phosphate of molybdenum, is referred to by

<sup>a</sup> Lindgren, Waldemar, and Ransome, F. L., *Geology and gold deposits of the Cripple Creek district Colorado*: U. S. Geol. Survey Prof. Paper 54, 1906, p. 124; also see pp. 12-13 of this report.

<sup>b</sup> Moissan, Henri, *Traité de chimie minérale*, t. 4, 1904, p. 685.

<sup>c</sup> Gin, Gustave, *A memoir on the methods of treatment of simple and complex ores of molybdenum, tungsten, uranium, and vanadium*: Trans. Am. Electrochem. Soc., vol. 12, 1907, p. 412.

<sup>d</sup> Des Cloizeaux, Alfred, *Manuel de mineralogie*, 1862, p. 271.



Hills,<sup>a</sup> who states that it was discovered by O. W. Knight, of Bangor, Me., in a molybdenite deposit at Catherine's Hill, of the same State.

Beside the foregoing minerals of molybdenum the presence of molybdenum in scheelite should be called to the reader's attention. Dana<sup>b</sup> by 17 analyses of scheelite from various parts of the world shows that molybdenum is present in this mineral in quantities varying from a trace up to about  $8\frac{1}{4}$  per cent.

### PHYSICAL PROPERTIES OF MOLYBDENUM.

Pure molybdenum is a white metal. Its appearance depends largely on the method of production: If obtained by reducing the oxides or the sulphides of molybdenum by hydrogen, it is a gray powder which under heat and pressure may be compacted into a metallic bar that is brittle and even fragile. Produced by aluminothermic methods or by reduction in the electric furnace, it is a compact metal, but owing to the absorption of carbon in the electric furnace, it is not pure and has different physical properties from the carbon-free metal, as is explained later. Pure compact molybdenum is malleable and is sufficiently soft to be filed and polished with ease. It will not scratch glass.

Within the last decade molybdenum was considered as absolutely lacking in ductility, but the research laboratory of the General Electric Co., at Schenectady, N. Y., has discovered a method of producing ductile forms of the element, which may be drawn into ribbons and fine wire.

Debray,<sup>c</sup> one of the earlier experimenters with the metal, considered it infusible, but although he failed to melt it, he discovered that at a high temperature it absorbed carbon. Moissan<sup>d</sup> subsequently obtained molten molybdenum in an electric furnace. Its exact melting point is still a matter of some question. The Bureau of Standards<sup>e</sup> has placed it about  $2,500^{\circ}\text{C.}$ , or  $4,500^{\circ}\text{F.}$  This is over  $1,400^{\circ}\text{C.}$  above the melting point of copper, about  $1,000^{\circ}\text{C.}$  higher than that of iron, and  $745^{\circ}$  above that of platinum. Osmium, tantalum, and tungsten are the only three metals listed by the Bureau of Standards as having higher melting points. A resumé of melting-point determinations of molybdenum is given by Pirani and Meyer.<sup>f</sup>

Moissan<sup>g</sup> determined the specific gravity of molybdenum as 9.01,<sup>h</sup> but recent research has shown that this value is low, and further that

<sup>a</sup> Hills, B. W., The molybdenite deposits of Tunk Pond, Me.: Mining World, vol. 31, 1909, p. 323.

<sup>b</sup> Dana, E. S., A system of mineralogy, 6th ed., 1911, p. 987.

<sup>c</sup> Moissan, Henri, *Traité de chimie minérale*, t. 4, 1904, p. 687.

<sup>d</sup> Moissan, Henri, *loc. cit.*

<sup>e</sup> Stratton, S. W., Melting points of chemical elements: Bureau of Standards Circular 35, 1912, 2d ed., p. 2.

<sup>f</sup> Pirani, M. V., and Meyer, A. R., Über den Schmelzpunkt des Wolframs und des Molybdäns: Ver. deut. Phys. Gesell., Jahrg. 14, 1912, pp. 426-428.

<sup>g</sup> Moissan, Henri, *loc. cit.*

<sup>h</sup> Moissan, Henri, *loc. cit.*

the specific gravity of the metal increases appreciably with the amount of mechanical working to which it is subjected, a phenomenon common to many other metals, such as copper, zinc, and tungsten.<sup>a</sup> The specific gravity of ductile molybdenum before drawing was determined in the research laboratory of the General Electric Co. as 10.02, and after drawing it ranged from 10.04 for a wire 3.75 mm. in diameter to 10.32 for a wire 0.038 mm. in diameter.

The accompanying table <sup>b</sup> shows the tensile strength of molybdenum wire of various diameters in comparison with that of tungsten wire of the same sizes and with that of one size of hard-drawn piano wire. It will be noted that the values for molybdenum are approximately one-half those for tungsten and for the steel wire of corresponding size; likewise, that the tensile strength of both molybdenum and tungsten increases very appreciably with the fineness of the wire. In other words, the more the metals are worked the stronger they become.

*Comparative tensile strength of molybdenum, tungsten, and steel wire.*

Wire.	Diameter in thou- sandths of an inch.	Diameter in mm.	Pounds per square inch.	Kilograms per square mm.
Molybdenum.....	5.0 2.8 1.5	0.125 .070 .038	200,000 to 260,000 230,000 to 270,000 270,000 to 310,000	140 to 182 161 to 189 189 to 217
Tungsten.....	5.0 2.8 1.5	.125 .070 .038	460,000 to 490,000 480,000 to 530,000 550,000 to 600,000	322 to 343 336 to 371 385 to 420
Hard-drawn piano wire.....	3.0	.075	507,000	356

The electrical resistance of ductile molybdenum is 5.6 microhms per cubic centimeter for hard-drawn wire and 4.8 for annealed wire, the resistivity being measured at 25° C.<sup>c</sup> The temperature coefficient of electrical resistance between 0° and 170° C. is 0.0050.<sup>d</sup>

The specific heat of molybdenum as determined by Defacqz and Guichard<sup>e</sup> is 0.072 at 93° C., 0.074 at 281° C., and 0.072 at 440° C.

The emissivity of molybdenum measured with red light having a wave length 0.650  $\mu$  has been determined by Burgess and Waltenberg<sup>f</sup> at temperatures of 2,000° and 2,500° C. as 0.43 and 0.40, by Coblentz<sup>g</sup> at room temperature as 0.51, and by Mendenhall and

<sup>a</sup> Fink, C. G., Ductile tungsten and molybdenum: Trans. Am. Electrochem. Soc., vol. 17, 1910, pp. 229-233.

<sup>b</sup> From Fink, C. G., loc. cit.

<sup>c</sup> Fink, C. G., loc. cit.

<sup>d</sup> Fink, C. G., loc. cit.

<sup>e</sup> Defacqz, E., and Guichard, M., Sur la détermination de la chaleur spécifique du tungstène et du molybdène: Anal. chim. Phys., ser. 7, t. 24, 1901, p. 139.

<sup>f</sup> Burgess, G. K., and Waltenberg, R. G., The emissivity of metals and oxides: Bureau of Standards Scientific Paper 242, 1914-15, p. 597.

<sup>g</sup> Coblentz, W. W., The reflecting power of various metals: Bureau of Standards Scientific Paper 152, 1911, p. 197.

Forsythe<sup>a</sup> at 1,000° and 2,400° C. as 0.44 and 0.37. Hasselberg<sup>b</sup> has determined the numerous lines of the spectrum of molybdenum.

As already stated, molybdenum produced by the reduction of molybdic oxide with carbon in an electric furnace does not possess the same physical properties as pure molybdenum, owing to its absorption of carbon. Metal obtained by this method is gray and brittle. It is also very hard and scratches steel and quartz; even the hardest file will not cut it when it contains a certain proportion of carbon. The melting point of the gray metal is much below that of pure molybdenum, and its specific gravity is also lower, ranging from 8.6 to 8.9, depending on the amount of carbon present. When pure molybdenum is surrounded with carbon and heated to about 1,500° C., it absorbs carbon and becomes hard. Inversely, if carbon-bearing molybdenum is melted with molybdenum dioxide, the carbon in the metal is oxidized and the molybdenum is refined and takes on the physical properties of the pure metal.<sup>c</sup>

### CHEMICAL PROPERTIES OF MOLYBDENUM.

Metallic molybdenum is only slowly oxidized at ordinary temperatures, and drawn molybdenum wire retains its luster almost indefinitely. On prolonged heating at a dull-red heat it becomes covered with a white coating of molybdic trioxide, and at 600° C. it oxidizes rapidly and the trioxide formed sublimes.

Fluorine attacks molybdenum at ordinary temperatures, chlorine at a dull-red heat, and bromine at a cherry red, and iodine does not attack it appreciably at temperatures as high as 700° to 800° C. Molybdenum is readily attacked by nitric acid but is not affected by hydrochloric acid, and sulphuric acid attacks it only when hot and concentrated. Fused alkalis act only slowly on the metal, but fused oxidizing salts, such as potassium nitrate, sodium peroxide, and potassium chlorate, attack it rapidly.

Molybdenum combines with oxygen to form several oxides, the three most stable of which are the sesquioxide ( $\text{Mo}_2\text{O}_3$ ), the dioxide ( $\text{MoO}_2$ ), and the trioxide ( $\text{MoO}_3$ ). The first two of these are basic, and few of their salts have been studied. The trioxide is acid. It is yellow when hot and white when cold. It is precipitated from solutions in hydrated form by nitric and hydrochloric acids, but is soluble in dilute sulphuric acid and the alkalis. When the hydrated oxide is gently ignited, it becomes dehydrated and much more difficultly soluble in dilute acids. When the trioxide is heated, it sublimes, sublimation commencing at 400° to 450° C.

<sup>a</sup> Mendenhall, C. E., and Forsythe, W. E., The relation between blackbody and true temperatures for tungsten, tantalum, molybdenum, and carbon, and the temperature variation of their reflective power: *Astrophys. Jour.*, vol. 37, 1913, pp. 380-390.

<sup>b</sup> Hasselberg, B., Die Spectra der Metalle im electrischen Flammenbogen; Spectrum des Molybdans: *K. Svenska vet. Acad. Handb.*, Ny Förl., Bd. 36, No. 2, 1902, p. 1.

<sup>c</sup> Moissan, Henri, *Traité de chimie minérale*, t. 4, 1904, p. 689.



A large number of elements combine with molybdic oxide to form molybdates. Of these; sodium molybdate,  $\text{Na}_2\text{MoO}_4$ , potassium molybdate,  $\text{K}_2\text{MoO}_4$ , and ammonium molybdate,  $(\text{NH}_4)_2\text{MoO}_4$ , are the chief soluble salts of molybdenum. The first two of these are formed by fusion of the trioxide with the requisite amount of sodium potassium carbonate. The latter is formed by the action of concentrated ammonia on the trioxide. The heavy metals, such as barium, calcium, and lead, form molybdates which are insoluble, and use of this fact is made in analytical work. Besides the normal molybdates of the form  $\text{X}'_2\text{MoO}_4$  many complex molybdates such as phospho-molybdates exist.

With the halogens molybdenum forms several compounds which vary greatly in stability. One compound,  $\text{MoO}_3 \cdot 2\text{HCl}$ , is of importance as it is volatile at  $250^\circ \text{C}$ . and its formation affords a means of separating molybdenum from tungsten.

Molybdenum forms two principal sulphides,  $\text{MoS}_2$  and  $\text{MoS}_3$ , both insoluble in dilute sulphuric and hydrochloric acids. These sulphides dissolve in the alkalis to form thiomolybdates of the order  $\text{X}'_2\text{MoS}_4$ , which react with acids to form sulphur, hydrogen sulphide, molybdenum disulphide, and molybdenum trisulphide.

Molybdenum combines with nitrogen, phosphorus, boron, carbon, and silicon. More complete references on the chemical properties of molybdenum are given by Roscoe and Schorlemmer,<sup>a</sup> who have been quoted freely in the foregoing discussion. Moissan<sup>b</sup> also gives detailed data on the chemistry of molybdenum.

### USES OF MOLYBDENUM.

The principal use of molybdenum is in the manufacture of alloy steels to which, particularly in conjunction with chromium, manganese, nickel, cobalt, tungsten, and vanadium, it imparts many desirable properties. These steels are used for a large variety of purposes such as for crank-shaft and propeller-shaft forgings, high-pressure boiler plate, guns of large bore, rifle barrels, armor plate, armor-piercing projectiles, permanent magnets, wire, and for self-hardening and high-speed machine tools. Metallic molybdenum is used in various electrical contact making and breaking devices in X-ray tubes, and in voltage rectifiers, and in the form of wire for filament supports in incandescent electric lamps, and for winding electric resistance furnaces, and in dentistry. Molybdenum is also employed in the manufacture of chemical reagents, dyes, glazes, disinfectants, etc.

<sup>a</sup> Roscoe, H. E., and Schorlemmer, C., *A treatise on chemistry*, vol. 2, 1913, pp. 1059-1082.

<sup>b</sup> Moissan, Henri, *Traité de chimie minérale*, t. 4, 1904, pp. 685-758.



## MOLYBDENUM STEELS.

The early experimenters with the use of molybdenum in steel were almost unanimous in condemning the metal. Thomas Blair, of Sheffield, England, in a pamphlet published in 1894,<sup>a</sup> quotes a case "where 1 per cent of molybdenum rendered good iron red-short, and utterly worthless." The poor results obtained by most of the pioneer investigators were due to two main causes—first, the molybdenum powder or ferromolybdenum used in making the steels was generally highly impure, containing sufficient amounts of sulphur, oxides, and other deleterious substances to spoil any steel. Second, the few molybdenum steels that were prepared in such a manner as to be free from objectionable impurities were generally ruined by improper heat treatment. It is, therefore, not surprising that at the outset the element received a "black eye," and this bad reputation has been an important factor in delaying a more general recognition of the value of molybdenum in the manufacture of special steels.

Thomas Swinden was probably the first to conduct a thorough investigation of a series of strictly comparable carbon-molybdenum steels ranging in composition up to 8 per cent molybdenum and 1.2 per cent carbon. Besides proving the extreme susceptibility of these steels to heat treatment and that they were easily ruined by even a short heating at 950° C. followed by a prolonged cooling, he showed that the molybdenum increased the tendency of the steel to harden on cold working, and that the effects of oil quenching, followed by tempering, were increased by its presence. Further, his investigation demonstrated that the effect produced by molybdenum depended to a large degree upon the treatment of the steels. With normalized steels, molybdenum considerably increased the tensile strength with only a slight reduction of ductility, and this influence was most marked in high-carbon steels. Hardened and tempered steels containing 1 to 2 per cent of molybdenum showed extremely high tenacity values, accompanied by high ductility, but when steels with higher percentages of molybdenum were hardened and tempered, they actually became inferior. Annealed steels diminished in strength and ductility as the molybdenum content increased, and this diminution was noticeable when 0.9 per cent or more of carbon was present. Throughout Swinden's tests the influence of carbon in the presence of molybdenum was marked, and there is no doubt that, like tungsten, molybdenum assists in producing hardness by helping to retain the carbon in solid solution. The results of Swinden's work appear in two excellent papers.<sup>b</sup> It suffices here to say that in a general

<sup>a</sup> Tungsten and chromium alloys.

<sup>b</sup> Swinden, T., Carbon molybdenum steels: Carnegie Scholarship Memoirs, Iron and Steel Inst. (London) vol. 3, 1911, pp. 66-124; and A study of the constitution of carbon-molybdenum steels, *ibid.*, vol. 5, 1913, pp. 100-168.

way molybdenum acts similarly to tungsten in steel, but that it is much more active—that is, less of it need be used to produce a given result, as might be suspected from its lower atomic weight (96) as compared with that of tungsten (184). It is impossible to give any absolute figures as to the relative influence of the two elements, as their effects are not exactly similar. Guillet<sup>a</sup> suggested that the effective ratio of molybdenum to tungsten was 1 to 4, but this is probably too high. Giesen<sup>b</sup> gives the ratio as 1 to 2.225, and from the tests correlated by Swinden it is seemingly between 1 to 2 and 1 to 3.

#### MANUFACTURE OF MOLYBDENUM STEELS.

The crucible process is generally used in the manufacture of molybdenum steel, and the molybdenum is added in the form of molybdenum powder, ferromolybdenum, or other molybdenum alloys. Molybdenum steel can, however, be made in the electric furnace, and also by the open-hearth process. It has also been produced on an experimental scale in the electric furnace directly from a mixture of hematite and molybdenite with coke, lime, and fluorspar.<sup>c</sup>

In adding molybdenum to steel it is generally advantageous to use ferromolybdenum, ordinarily containing about 80 per cent Mo, as the melting point of this alloy is several hundred degrees centigrade lower than that of the ordinary commercial brands of powdered molybdenum. Further, powdered molybdenum seemingly is more prone to oxidation than the ferro-alloy, but on the other hand it generally contains 4 or 5 per cent of various oxides of molybdenum which may aid in removing excess carbon.

As molybdenum is generally used in steel in conjunction with chromium, tungsten, nickel, vanadium, etc., it is often added in the form of alloys with these metals. Standard alloys of this type on the market are chrome-molybdenum, with 50 per cent molybdenum and 50 per cent chromium; molybdenum-nickel, with 75 per cent molybdenum and 25 per cent nickel; and ferromolybdenum-tungsten, containing molybdenum and tungsten in the proportion of 3 to 1. From 1 to 7 per cent vanadium, according to specifications, is sometimes added to the last-mentioned alloy.

#### USE OF MOLYBDENUM IN TOOL STEELS.

Molybdenum acts similarly to tungsten in the presence of chromium and manganese, and in combination with either of these elements and carbon it produces a self-hardening steel which is said to be a

<sup>a</sup> Guillet, Leon, *Revue de Métallurgie*, 1904, p. 390.

<sup>b</sup> Giesen, Walter, *The special steels in theory and practice: Carnegie Scholarship Memoirs, Iron and Steel Inst. (London), vol. 1, 1909, p. 31.*

<sup>c</sup> Keeney, R. M., *The production of steels and ferroalloys directly from ore in the electric furnace: Carnegie Scholarship Memoirs, Iron and Steel Inst. (London), vol. 4, 1912, pp. 108-184.* Describes production of molybdenum steel (pp. 173-175).

little tougher than the corresponding tungsten steel. A typical steel of this kind is one containing 4 to 6 per cent molybdenum, 1 to 2 per cent chromium, and 1.85 per cent carbon. Molybdenum likewise has the same properties as tungsten with reference to producing the high-speed qualities that enable a steel to retain its temper and hardness at a red heat. These qualities are developed by cooling the steel moderately fast from a high temperature, treatment that prevents the usual critical changes and keeps the steel in the austenitic condition. It is believed by many that high-speed steels produced with molybdenum are superior to the corresponding tungsten steels, both as regards toughness and durability. It is stated that they take a fine cutting edge. According to Gledhill,<sup>a</sup> one of the qualities of molybdenum high-speed steel is that it does not require such a high temperature in hardening as does tungsten steel, and he states that if a temperature of 1,000° C. is exceeded, the tools made from the steels are inferior and their life shortened. Carpenter<sup>b</sup> states, on the other hand, that molybdenum steels should be heated to 1,000° to 1,100° C. before they are cooled, whereas tungsten steel must be heated to about 1,200° C. The superior toughness of molybdenum high-speed steels is attributed to the fact that they contain more iron, as less molybdenum need be used to obtain the same result as with tungsten, and also because a lower heat is required in tempering them.

Certain difficulties, however, have been encountered in the use of molybdenum tool steels. Some users say that they are liable to crack in quenching, and others that they do not hold their cutting edge after retreatment as well as before. This deterioration in the steel upon repeated heating for dressing and treatment has been ascribed to the disappearance of molybdenum from the outer skin of the steel through volatilization. In a few instances service tests with these steels have shown irregular cutting speeds and have indicated a tendency of the molybdenum to render the tools brittle and weak in their bodies. Some users have found that molybdenum tool steel was apt to be seamy and to contain physical imperfections, also that it is apt to fire-crack during treatment. Molybdenum tool steels of high carbon content require great skill in their preparation owing to the difficulty of judging by color the definite temperature required for hardening. Further, great care in annealing molybdenum steel is necessary after it has been worked into bars and before it is cut into shapes for tools, and previous to hardening.

Some of the objectionable features mentioned above, and perhaps most of them, are due to the use of impure ingredients in the manufacture of the steels or to improper heat treatment, and undoubtedly

<sup>a</sup> Gledhill, J. M., The development and use of high-speed tool steel: Jour. Iron and Steel Inst., No. 11, 1904, pt. 2, pp. 127-182.

<sup>b</sup> Carpenter, High-speed tool steels: Jour. Iron and Steel Inst., No. 1, 1905, p. 460.



many of them can be overcome. However, most manufacturers, particularly in this country, have almost entirely discontinued the use of molybdenum as a major constituent in tool steel and are now using it largely in a minor capacity in conjunction with tungsten, cobalt, etc. Used in this way many of the difficulties mentioned above have been overcome, and the resulting steels have at the same time required certain superior qualities imparted by the molybdenum. A characteristic steel of this type is one with 16 to 18 per cent tungsten, 1.5 to 2 per cent molybdenum, 4 to 4.5 per cent chromium, and 0.6 per cent carbon, and it is said to be superior in cutting efficiency to the corresponding tungsten-chromium steel. It is also stated that the addition of the molybdenum in this steel results in a finer texture and lessens the chances of injuring the steel previous to hardening. The addition of molybdenum is particularly common in high-speed steels where cobalt is used in conjunction with tungsten. This type of steel contains about 16 to 18 per cent tungsten, 4 to 5.5 per cent cobalt, and 0.25 to 1.5 per cent molybdenum. Small percentages of vanadium are sometimes used in connection with molybdenum and tungsten in tool steels and are said to increase their strength and cutting efficiency. An analysis of such a steel containing both vanadium and cobalt in conjunction with tungsten, chromium, and molybdenum, and known as iridium steel, follows:

*Approximate average analysis of iridium steel.<sup>a</sup>*

Constituent.	Per cent.	Constituent.	Per cent.
Cobalt.....	4.25	Manganese.....	} Low.
Tungsten.....	16.00	Silicon.....	
Chromium.....	3.55	Sulphur.....	
Vanadium.....	.67	Phosphorus.....	
Molybdenum.....	.80	Carbon.....	0.60
		Iron.....	b 74.00

It is stated that this iridium steel will do 60 per cent more work than the best tungsten steel.

The introduction of high-speed tool steels has completely revolutionized machine-shop practice. Gledhill<sup>c</sup> states that, as compared with cutting speeds of 30 to 50 feet per minute for ordinary crucible steel, high-speed steels are now made that will cut 500 feet per minute on steel, and drill cast iron at the rate of 25 inches per minute. Planing speeds of 15 to 25 feet per minute have been increased to 50 and 60 and even to 80 feet per minute. The high-speed twist drill has also made radical changes in practice. To-day it is customary to stack small plates and drill them instead of punching them, not only saving time, but avoiding the liability of injuring the

<sup>a</sup> Hess, F. L., Cobalt, Mineral Resources, U. S., 1912, U. S. Geol. Survey, 1913, p. 967.

<sup>b</sup> Approximate.

<sup>c</sup> Gledhill, J. M., The development and use of high-speed tool steel: Jour. Iron and Steel Inst., No. 11, 1904, pp. 127-182.



metal by straining it. The plates for torpedo boats are now drilled instead of punched. It was formerly the custom to core in the holes in pipe flanges, but they are now drilled.

#### USE OF MOLYBDENUM IN MAGNET STEEL.

The molybdenum steels used in making permanent magnets are similar in composition to the high-speed molybdenum steels. They generally contain 2 to 3 per cent molybdenum, 0.5 to 0.7 per cent carbon, and sometimes about 0.5 per cent chromium. After hardening they retain their magnetism longer than hardened carbon steel, and are said to be superior to tungsten magnet steels. Some steels of this type contain up to 6 per cent molybdenum.

#### USE OF MOLYBDENUM IN ACID-RESISTING STEELS.

Alloyed with chromium, and sometimes with chromium and tungsten, molybdenum is used in preparing acid-resisting steels. Borchers<sup>a</sup> states that an alloy containing 2 to 5 per cent molybdenum, about 10 per cent chromium, and a little or no carbon, is practically acid proof. Molybdenum-tungsten-chromium-iron alloys have also been made that are insoluble in hydrochloric, sulphuric, or nitric acid, and an alloy containing 60 per cent chromium, 35 per cent iron, and 2 to 3 per cent molybdenum is said to resist even the action of boiling aqua regia.<sup>b</sup> Differences in the heat treatment of these alloys have a great effect on their acid-resisting qualities.

#### USE OF MOLYBDENUM IN OTHER ALLOY STEELS.

Molybdenum is used in conjunction with chromium and nickel to produce steels with a wider heat-treatment range and with even greater tensile strength than are possessed by chrome-nickel steel. These chrome-nickel-molybdenum steels contain about 1 per cent chromium, 2 to 3 per cent nickel, and 0.25 to 0.5 per cent molybdenum, and on account of their high tensile strength and elastic limit, they have a range of usefulness that is limited only by their cost. These properties make them of special value for crank shafts, propeller shafts, and other machine parts that are subjected to alternating and repeated stresses. For example, they are used to some extent in the frames and axles of automobiles and railroad rolling stock. The high tensile strength of these steels is also responsible for their use in the best grades of high-pressure boiler plate such as is used on torpedo boats. They are also employed in making guns of large bore, and rifle barrels, and for such use they have the added advantage of being highly resistant to the erosive action of the gases generated by the explosives.

<sup>a</sup> Borchers, W., *Acid-resisting alloys*; Engineer (London), vol. 114, July 26, 1912, p. 83.

<sup>b</sup> Benner, R. C., *Recent advances in industrial chemistry*; Min. and Sci. Press, vol. 105, 1912, p. 629.

It has been found that the addition of a small percentage of molybdenum is advantageous in the manufacture of armor plate and shields for small guns. The molybdenum is said to make the steel softer when it is annealed so that it can be planed more readily, and upon hardening, the molybdenum content produces an increased tensile strength. The exact compositions of the various molybdenum steels used in armor plate are, of course, carefully guarded secrets. Stoughton <sup>a</sup> states that Krupp armor plate (probably of early manufacture) contains about 3.25 per cent nickel, 1.5 per cent chromium, and 0.25 per cent carbon. The assumption is safe that the introduction of 0.25 to 0.5 per cent of molybdenum in a steel of this composition would increase both its hardness and its tensile strength.

The use of molybdenum in connection with chromium is reported for producing steels of great hardness and high elastic limit, which are employed in the manufacture of armor-piercing projectiles. About 1 per cent of molybdenum added to a nickel steel is said to impart to it remarkable drawing qualities.

#### FERROMOLYBDENUM.

Ferromolybdenum has been used for electrodes in arc lamps.<sup>b</sup>

#### USE OF MOLYBDENUM IN STELLITE.

Small percentages of molybdenum are used in certain patented non-ferrous alloys consisting essentially of chromium and cobalt. These alloys are known under the trade name of "Stellite," and possess remarkable high-speed qualities when used for machine tools. They are also employed for cold chisels, wood-working tools, cutlery, etc. Their use in cutlery is of particular interest as they do not tarnish under atmospheric influences and are unaffected by fruit acids.

In so far as molybdenum is concerned, two stellite alloys are of interest, namely, those containing cobalt, chromium, and molybdenum, and those containing cobalt, chromium, tungsten, and molybdenum. Haynes,<sup>c</sup> the inventor of these alloys, says:

When molybdenum is added to a 15 per cent cobalt-chromium alloy [15 per cent chromium], the alloy rapidly hardens as the molybdenum content increases, until the content of the latter metal reaches 40 per cent, when the alloy becomes exceedingly hard and brittle. It cuts keenly and deeply into glass, and scratches quartz crystal with ease. It takes a beautiful polish, which it retains under all conditions, and on account of its extreme hardness its surface is not readily scratched. When 25 per cent of molybdenum is added to a 15 per cent chromium alloy, a fine-grained metal results, which scratches glass somewhat readily, and takes a strong, keen edge. Its color and luster are magnificent, and it will doubtless find a wide application for fine, hard cutlery. It can not be forged, but casts readily, and its melting point is not abnormally high.

<sup>a</sup> Stoughton, Bradley, *The metallurgy of iron and steel*, 1908, p. 407.

<sup>b</sup> Editorial, *Molybdenite*: Min. and Sci. Press, vol. 104, 1912, p. 781.

<sup>c</sup> Haynes, Elwood, *Alloys of cobalt with chromium and other metals*: Trans. Am. Inst. Min. Eng., vol. 44, 1912, pp. 576-577.

These stellite alloys are silvery-white. They are insoluble in nitric acid and are only slowly attacked by hydrochloric acid. Following is the analysis of a typical molybdenum stellite.

*Analysis of a typical molybdenum stellite.<sup>a</sup>*

Constituent.	Per cent.
Mo.....	22. 50
Co.....	59. 50
Cr.....	10. 77
Fe.....	3. 11
Mn.....	2. 04
C.....	. 87
Si.....	. 77
S.....	. 084
P.....	. 040
W.....	. 00
Ni.....	. 00

---

99. 684

### USES OF METALLIC MOLYBDENUM.

The uses of metallic molybdenum have been greatly extended by the discovery of certain processes by which it can be obtained in ductile form.<sup>b</sup> As wire, it is used for supporting the filaments in incandescent electric lamps. Of course, the quantity of the element used in a single lamp of ordinary candlepower is extremely small, but in argon-nitrogen-tungsten lamps of high candlepower the molybdenum supports are of very appreciable size and weight. Molybdenum wire is also employed for winding electric resistance furnaces, and for this use it has proved both cheaper and superior to platinum because of the quicker heating and the higher temperatures attainable.<sup>c</sup> In this work it must, of course, be protected from the air to prevent its oxidation.

Molybdenum has been successfully substituted for platinum and for platinum iridium in various electric contact making and breaking devices. On account of its high heat conductivity in the ductile form, and because its relative cheapness permits its use in large masses, the formation of heavy coatings of nonconducting oxide are prevented, and under the conditions existing in these contacts, any thin coatings of oxide formed are conductors. The Coolidge X-ray tube and a voltage rectifier of recent manufacture have a very considerable proportion of ductile molybdenum used in their construction. Gold-covered molybdenum wire is used in dentistry to a considerable extent, and research work is being carried on to extend the use of special molybdenum alloys in this field.<sup>d</sup>

<sup>a</sup> Hibbard, H. D., Manufacture and uses of alloy steels: Bull. 100, Bureau of Mines, 1915, p. 61.

<sup>b</sup> See Coolidge, W. D., U. S. Patent 1082933, Dec. 30, 1913.

<sup>c</sup> Winne, R., and Dantsizen, C., Small electric furnace with heating element of ductile tungsten or ductile molybdenum: Jour. Ind. and Eng. Chem., vol. 3, October, 1911, p. 770.

<sup>d</sup> Fahrenwald, F. A., A development of practical substitutes for platinum and its alloys, with special reference to alloys of tungsten and molybdenum: Am. Inst. Min. Eng., Bull. 109, January, 1916, pp. 103-149.



It is suggested that molybdenum may be successfully used with tungsten as a thermocouple for the measurement of high temperatures. The advantage in the use of these metals is that they have much higher fusing points than platinum, platinum-iridium, or platinum-rhodium couples, such as are ordinarily used, and although they oxidize at high temperatures, they might be protected from the air by covering them with fused magnesia or similar substances. The voltage curve of the molybdenum-tungsten couple has been studied by Northrup.<sup>a</sup>

#### USE OF MOLYBDENUM IN CHEMICALS.

The chief use of molybdenum in chemicals is as ammonium molybdate,  $(\text{NH}_4)_2\text{MoO}_4$ , and sodium molybdate,  $\text{Na}_2\text{MoO}_4$ . The first is employed principally as a reagent in the quantitative determination of phosphorus in iron and steel, various ores, fertilizers, soils, etc., and in the quantitative determination of lead. The equivalent of several tons of metallic molybdenum is said to be consumed annually in this form by the iron and steel plants in the United States for the determination of the phosphorus content of their products. It is reported that ammonium molybdate is also used for fireproofing, and as a disinfectant for cloth used in railroad coaches and for similar purposes. It is said to be a strong germicide. Sodium molybdate was at one time used rather extensively in ceramics to impart a blue color to pottery and in the manufacture of certain glazes, but it is understood to have been replaced almost wholly in this field by cobalt compounds. It is also used in dyeing silks and wools. Molybdenum indigo,  $\text{Mo}_5\text{O}_7$ , is used for coloring rubber and is said to be noninjurious to the material.<sup>b</sup>

Fast colors in a large variety of shades may be produced on leather by employing molybdenum tannate in conjunction with logwood extracts.<sup>c</sup>

A process involving the use of molybdenum for the preservation of cordite in hot climates is said to have been discovered in France.<sup>d</sup> It is also rumored that molybdenum is used in the synthetic production of ammonia under processes covered by German patents and in which it is said to act as a "promoter" to the catalyzers used. It is well known that ammonia may be made by heating molybdenum oxide or molybdenum hydroxide, or mixtures of the two, to

<sup>a</sup> Northrup, E. F., Tungsten and molybdenum; their thermal E. M. F.: *Met. and Chem. Eng.*, vol. 11, January, 1913, p. 45.

<sup>b</sup> Hess, F. L., Tungsten, nickel, cobalt, etc.: *Mineral Resources U. S. for 1908*, U. S. Geol. Survey, 1909, p. 746.

<sup>c</sup> Pozzi-Escot, E., Production de couleurs fixes sur tous genres de cuirs, par l'emploi de sels de molybdène combinés à des matières tannantes ou à des couleurs mordantes végétales: *Compt. rend.*, t. 135, 1902, p. 801.

<sup>d</sup> Editorial, Molybdenite: *Min. and Sci. Press*, vol. 108, 1914, p. 860.



about 500° to 600° C. with equal equal parts of nitrogen and hydrogen, and under a pressure of about 60 atmospheres, to form molybdenum nitride, which when heated in a vacuum yields ammonia and metallic molybdenum.<sup>a</sup>

### PRODUCTION OF MOLYBDENUM ORES.

Queensland, New South Wales, and Norway have to date furnished the major part of the world's production of molybdenum, the output derived from all other countries, including the United States, probably not amounting to 10 per cent of the total. Austria, Canada, France, Germany, Great Britain, Japan, Mexico, Natal, Russia, Sweden, the United States, and Australian States other than those mentioned above have at times made a small production of molybdenum ore, but with the exception of the United States their output has been too small to be worthy of consideration.

The following table shows the tonnage and value of high-grade molybdenite concentrates produced in Queensland, New South Wales, and Norway from 1902 to 1914:

*Production of high-grade molybdenite concentrates in Queensland, New South Wales, and Norway, 1902 to 1914.*

Year.	Queensland, <i>a</i>		New South Wales, <i>b</i>		Norway, <i>c</i>	
	Short tons.	Dollars.	Short tons.	Dollars.	Short tons.	Dollars.
1902.....	<i>d</i> 45.9	<i>d</i> 26,770	16.8	8,960	22	16,100
1903.....	<i>d</i> 26.9	<i>d</i> 10,220	32.5	21,690	34	21,400
1904.....	23.6	13,010	28.3	13,270	33	17,400
1905.....	70.8	41,340	21.7	12,200	51	16,300
1906.....	118.9	74,330	36.6	23,350	<i>e</i> 1,129	14,200
1907.....	74.0	41,080	24.2	17,340	33	12,900
1908.....	98.7	44,960	9.5	4,520	39	13,400
1909.....	103.9	45,120	31.5	15,810	33	12,100
1910.....	118.6	58,640	53.2	27,580	.....	.....
1911.....	111.4	64,610	23.1	12,610	2	800
1912.....	114.6	84,420	63.3	18,030	23	5,400
1913.....	74.3	92,460	88.3	33,100	13	3,200
1914.....	87.1	185,830	68.8	55,720	.....	.....

*a* Data from annual reports of the under secretary of mines, Queensland.

*b* Data on quantity exported from annual report, Department of Mines, New South Wales, 1914.

*c* Data from Norges Bergvaerkdrift, Norges Officielle Statistik.

*d* The molybdenite production of Queensland in 1902 and 1903 as officially reported includes a small tonnage of bismuth and tungsten ore.

*e* Probably ore.

### QUEENSLAND.

The first official record of a production of molybdenite in Queensland was in 1900, when the output amounted to 12.3 short tons of high-grade material. The production gradually increased to 118.9 short tons in 1906, and it varied little from this figure until 1913, when it dropped to 74.3 tons, but rose again in 1914 to 87.1 tons. The

*a* Tucker, S. A., and Moody, H. R., The production of hitherto unknown metallic borides: Jour. Chem. Soc., vol. 81, 1902, p. 16.

value of the production in the latter year, however, showed an enormous increase, owing to an advance in prices, and amounted to \$185,830, or more than twice the value of the 1913 production, which was only slightly less in tonnage. The total production to the end of 1914 was approximately 1,030 short tons, valued at about \$760,000. The bulk of this material was mined at Wolfram Camp, in the Chillagoe field, about 120 miles southwest from Cairns, in northern Queensland. The mines at Bamford, in the same field, are, however, credited with a small production. In 1914 the principal producing properties in the Wolfram Camp district were the Murphy & Geaney mine, which treated 1,233 long tons of ore, which yielded 20.2 tons of molybdenite, valued at £9,543, and 83.4 tons of bismuth-tungsten concentrates, valued at £5,558, and the Larkin & Wade mine, which made a total output of 1,254 long tons of milling ore, which yielded 9.65 tons of molybdenite and 54.55 tons of bismuth-tungsten concentrates. Other mines at Wolfram Camp which made smaller productions of molybdenite were the Gillian Lease, Tulley, Tulley Block, Hillside United, Smith United, and Mulligan & McIntyre. At Bamford the Trafalgar, Evening Star, Northern United, Morning Star, and Sunny Corner mines made a total output of 8.55 long tons of high-grade molybdenite.

The total production of the Chillagoe district in 1914 was 76.25 long tons of molybdenite. The remainder of the output came from the Star River (Ollera) district and from a newly discovered field at Glen Atherton; on Upper Tinaroo Creek, in the Gladstone district.

Detailed descriptions regarding the occurrence and production of molybdenite in Queensland are given in the annual reports of the acting undersecretary for mines and in the references following:

*Literature on occurrence and production of molybdenite.*

Saint-Smith, E. C., Molybdenite in the Stanthorpe-Ballandean districts, Southern Queensland: Queens. Govt. Min. Jour., vol. 15, 1914, pp. 184-189.

Cameron, W. E., Wolfram, molybdenite, and bismuth mining at Wolfram Camp, Hodgkinson goldfield: Queens. Govt. Min. Jour., vol. 4, 1903, pp. 350-352.

Cameron, W. E., Wolfram and molybdenite mining: Queens. Govt. Min. Jour., vol. 5, 1904, pp. 62-65.

Ball, L. C., The wolfram, molybdenite, and bismuth mines of Bamford: Geol. Surv. Pub. 248, Mines Dept., Queensland.

Ball, L. C., Rare-metal mining in Queensland—résumé of recent field studies: Queens. Govt. Min. Jour., vol. 14, pp. 4-7.

### NEW SOUTH WALES.

The production of molybdenite in New South Wales was first reported in 1902. In that year the output was 16.8 short tons, valued at \$8,960. The total production to the end of 1914 was 498 short tons, valued at over \$264,000.

The principal producing molybdenite mines are at Whipstick in the Pambula division, at Kingsgate in the Glenn Innes division, and near Deepwater in the Deepwater division. The annual report of the Department of Mines for the year 1914 states that at Whipstick the Whipstick Mines (Ltd.) raised 63 long tons of ore, estimated to contain 9 tons of concentrates, valued at £5,000. At Kingsgate, in the Glenn Innes division, the Yates mine produced 8.75 tons of concentrates, containing 94 per cent molybdenite and valued at £4,700, and the Glenn Innes Molybdenite and Bismuth Syndicate sold 1.6 long tons of concentrates containing 92 per cent molybdenite for £655. The Sacks mine at Kingsgate produced 0.85 ton of molybdenite, valued at £340. In the Deepwater division the Bow Creek Molybdenite Mines (Ltd.) produced 65 long tons of ore, which yielded 1.63 tons of concentrates, valued at £735. In the same division 11 tons of ore, which yielded 1.5 tons of molybdenite concentrates, valued at £700, was mined by E. A. Baker, and 28 tons of ore, which yielded 1.4 tons of molybdenite, assaying 93 per cent  $\text{MoS}_2$ , and valued at £667, was mined at Four Mile by E. L. Key. At Rocky River, in the Tantafield division, 80 tons of molybdenite ore, valued at £631, was mined, and in the Bathhurst division, 38 tons, valued at £800. The table on page 29 shows that the greatest production of molybdenite in New South Wales was in 1913, when 88 short tons, valued at \$33,100, was mined, and that in 1914 the output decreased to 68.8 tons. On account of the marked advance in prices, however, the value of the output in the latter year was more than \$20,000 in excess of the value of the production in 1913.

Detailed descriptions of the occurrence and production of molybdenite in New South Wales are given in the annual reports of the department of mines, New South Wales, and by Andrews.<sup>a</sup>

#### NORWAY.

In Norway the production of high-grade molybdenite concentrates has averaged about 30 tons per annum since 1902. The table on page 29, in which the output of this country is given by years, shows that in 1906 a production of 1,129 short tons was reported. This figure doubtless does not represent the output of a high-grade molybdenite concentrate, but rather the production of ore in that year.

The principal molybdenite-producing sections in Norway are the Provinces of Lister og Mandal and Nedenes, on the extreme southern end of the peninsula. The district of Fjotland, in the former Province, is probably richer in molybdenite than any yet discovered in Norway. A mine at Knaben, in this district, has been the largest

<sup>a</sup> Andrews, E. C., Molybdenum: New South Wales Dept. of Mines and Agriculture, Geol. Survey Min. Resources, No. 11, 1906, 17 p.



and probably the only successful producer in Norway. This mine, owned by George G. Blackwell & Sons Co., of Liverpool, England, has made an average output of about 25 short tons per annum. In 1912 about 200 tons of low-grade molybdenite ore was mined at Reinshommen, in Fjotland, and about 610 tons of ore averaging about 2.4 per cent  $\text{MoS}_2$  at Langvand. Of this latter amount 390.5 tons was milled, yielding 0.5 metric ton of concentrates, assaying 51 per cent of  $\text{MoS}_2$ , and 10 tons of second concentrates running about 23 per cent  $\text{MoS}_2$ .<sup>a</sup>

It is reported that on account of the increased demand and high prices being paid for molybdenum a considerable interest is being taken at the present time in developing some of the numerous Norwegian deposits. The Ore Concentration Co. (Ltd.), of London, has recently tested several parcels of Norwegian molybdenite ore aggregating over 100 tons. This company states that the Elmore oil-flotation process is in use at two mines in Norway, and that large quantities of molybdenite concentrates have already been shipped.<sup>b</sup>

#### UNITED STATES.

The principal production of molybdenum ore in the United States was in 1903, when, according to Pratt,<sup>c</sup> 795 short tons of wulfenite and molybdenite concentrates, valued at \$60,865, was marketed. Probably 750 tons, or more, of this material consisted of wulfenite concentrates obtained by sluicing tailings from a cyanide mill at Mammoth, Ariz. The ore from which these tailings were derived came from the Mammoth mine at Schultz, about 3 miles away. The remainder of the output, consisting of molybdenite concentrates, was probably derived from the Crown Point mine, Chelan County, Wash., with perhaps a ton or two from the mine of the American Molybdenum Co., at Cooper, Me., and from other sources. Previous to 1903 practically the entire commercial output of molybdenum ore in this country consisted of 20 to 30 tons of high-grade molybdenite obtained from the Crown Point mine.<sup>d</sup> From 1903 to 1914 the United States Geological Survey has recorded a production of a marketable molybdenum product in only three years, namely, 1905, 1906, and 1907. The output in these three years was confined to small lots of molybdenite derived largely from the Crown Point mine and from a deposit near Homestake, Mont., and to a few tons of wulfenite concentrates

<sup>a</sup> Editorial, Norwegian molybdenite: Eng. and Min. Jour., vol. 98, 1914, p. 820.

<sup>b</sup> Editorial, Molybdenum recovery by the Elmore process: Eng. and Min. Jour., vol. 99, 1915, p. 907.

<sup>c</sup> Pratt, J. H., The steel-hardening metals: Mineral Resources U. S. for 1903, U. S. Geol. Survey, 1904, p. 308.

<sup>d</sup> Pratt, J. H., Tungsten, molybdenum, uranium, and vanadium: Mineral Resources U. S. for 1901, p. 268, and for 1902, p. 287, U. S. Geol. Survey, 1902 and 1903.



from Arizona.<sup>a</sup> Official statistics as to the production in 1914 are as yet not available, but it is known to have been small and to have consisted for the most part of wulfenite concentrates obtained from a small mill erected by F. H. Hereford and R. O. Boykin, of Tucson, Ariz., to re-treat the tailings piles already mentioned at Mammoth, Ariz. Small lots of high-grade molybdenite coming from the Crown Point mine, Chelan County, Wash., and the Romero Mining Co.'s property near Porvenir, N. Mex., were also marketed in 1914.

Since January, 1915, the production of both molybdenite and wulfenite in this country has shown a notable increase. In October, 1914, the Primos Chemical Co., of Primos, Pa., purchased a molybdenite prospect on Red Mountain, near Empire, Colo., and developed the property to such an extent that shipments of ore were made in January, 1915, and it is understood that they continued at the rate of two to three 25-ton carloads per week until the mine was shut down for the winter. It is reported that the ore shipped contained 2 to 3 per cent of molybdenite. Since the beginning of 1915 an assaying and ore-testing firm of Denver, Colo., has purchased considerable quantities of molybdenite ore coming from various mines in this country and Canada, and is said to have marketed a small tonnage of molybdenite concentrates of good grade derived from these ores. The embargo placed by Great Britain on the exportation of molybdenum ore from her colonies has of course since prevented further receipt of Canadian ores by this company. It is reported that a contract was placed with another company of Denver, Colo., in 1915, for the concentration of 500 tons of low-grade molybdenite ore from a deposit on Bartlett Mountain, Summit County, Colo., and, further, that several hundred tons of low-grade ore from neighboring claims were concentrated by flotation with fair results at Leadville, Colo. Considerable development work was done on these deposits during the year. Noteworthy development, resulting in the production of a considerable tonnage of low-grade ore, was likewise done by the Leviathan Mines Co. on its molybdenite property near Copperville, Mohave County, Ariz. Several molybdenite prospects throughout the West produced small quantities of ore, and small lots of high-grade molybdenite were obtained from some of them by careful selective mining on streaks of high-grade material and by hand picking.

The production of wulfenite also increased. F. H. Hereford and R. O. Boykin enlarged their mill at Mammoth, Ariz., to over twice

---

<sup>a</sup> Pratt, J. H., Steel and iron hardening metals: Mineral Resources U. S. for 1905, U. S. Geol. Survey, 1906, p. 412. Hess, F. L., Nickel, cobalt, vanadium, etc.: Mineral Resources U. S. for 1906, U. S. Geol. Survey, 1907, p. 539. Hess, F. L., Tungsten, nickel, cobalt, etc.: Mineral Resources U. S. for 1907, U. S. Geol. Survey, 1908, p. 721.

its previous capacity, and during the summer were producing a wulfenite concentrate containing approximately 22 per cent  $\text{MoO}_3$  at the rate of about 2 tons per day.

A small production of wulfenite concentrates were said to have been derived in 1915 by Mr. Frank Widener, of Cutter, N. Mex., from a deposit about 30 miles west of that place. Considerable development work was done on the Old Yuma mine near Tucson, Ariz., and a small mill was erected for the treatment of the wulfenite ore by Col. Epes Randolph and associates of Tucson.

The author believes that the output of molybdenum contained in the molybdenite and wulfenite ores mined in the United States in 1915 will compare favorably with that of any other country. Whether it will continue to increase in the future depends entirely on the demand.

#### CANADA.

The production of molybdenum ore in Canada has to date been confined to small lots of high-grade molybdenite obtained by cobbing and hand-picking ore from rich streaks in certain deposits in the Provinces of Ontario and British Columbia. It is reported that these lots, which have usually averaged 500 to 2,000 pounds in weight, have been largely sold to chemical manufacturers in the United States, although a few of them have been shipped to England. In addition to this output of high-grade material perhaps 100 to 200 tons of medium-grade molybdenite ore has been shipped to the United States or to England for concentration. Of course, since the embargo placed by Great Britain on the exportation of molybdenum ore from her colonies such shipments to the United States have ceased. Although up to the present time no production worthy of mention has been made in Canada, that country has several promising deposits of high-grade molybdenite ore and is doubtless capable of a considerable production.

#### IMPORTS OF MOLYBDENUM.

Under the tariff of 1909 the duty on molybdenum or ferromolybdenum valued at \$200 per ton or less was 25 per cent ad valorem, and on material valued at more than \$200 per ton it was 20 per cent ad valorem. Under the existing tariff this duty was reduced to 15 per cent ad valorem, irrespective of value. Ores of molybdenum may be imported free. Although it is known that small lots of high-grade molybdenite have been brought into this country at frequent intervals, no figures as to the quantity so imported can be given, as no separate account of the imports of molybdenum ore are kept by the customs service. However, the total quantity is probably not large, as practically the entire output of Queensland, New

South Wales, and Norway, which have been the principal producing countries to date, has been taken by Germany, England, and France. It is understood that during the early part of 1915 about 60 tons of high-grade molybdenite ore was imported from Canada for concentration in this country; but the action of the British Government in placing molybdenum ores, metallic molybdenum, and ferromolybdenum on the list of absolute contraband has made impossible further shipments of such material into this country, either from Canada or other British possessions.

The table following shows the amount and value of the molybdenum and ferromolybdenum imported into the United States during the years 1911 to 1915. Previous to 1911 imports of these metals were not separately reported by the customs authorities. It will be noted that during the period covered by the table imports of these metals were practically negligible.

*Quantity and value of imports of molybdenum and ferromolybdenum into the United States, 1911 to 1915.*

Calendar year.	Tons.	Dollars.
1911.....	8.5	11,409
1912.....	4.0	4,670
1913.....	7.0	15,939
1914.....	.1	59
1915.....	2.0	2,370

### MARKET FOR MOLYBDENUM.

Under normal conditions practically 90 per cent of the world's molybdenum production is used in making alloy steel and only about 10 per cent in the manufacture of chemicals and for all other purposes. At first glance it is therefore somewhat surprising that with the greatly increased demand for alloy steels occasioned by the European war the market for the element should not have developed to a larger degree, and in a greater measure kept pace with the phenomenal demand for its sister metal, tungsten, especially as molybdenum has in the past been used principally in the manufacture of field artillery, coast-defense and naval guns, rifle barrels, armor plate, armor-piercing projectiles, etc., and of high-speed tool steel, which is finding such a large use in the manufacture of these and other munitions. The ultimate market for molybdenum has, however, always been largely in Europe, the use of the element in steel being much more popular abroad than in the United States. Up to the present time France, England, and Germany have taken practically the entire molybdenum output of the world, and in comparison an almost negligible quantity of molybdenite and wulfenite, derived from the small domestic production and small quantities of molybdenite imported, have been treated in this country. Outside



of a few companies that are purchasing high-grade molybdenite concentrates for the manufacture of chemicals, there are even to-day only five concerns in the United States that are making metallic molybdenum or ferromolybdenum. Of course, there are many ore buyers in this country who are in the market for molybdenite concentrates, but their market is in turn almost wholly abroad, and they act either as middlemen, on their own behalf, or as agents for European purchasers.

The development of the molybdenum market has in the past and is at present being retarded by the fact that manufacturers who might use or might investigate the possibilities of using the metal are kept out of the market by the fear of not being able to obtain steady supplies. On the other hand, those who might be interested in the development of some of the extensive low-grade molybdenite or wulfenite deposits in this country are prevented from doing so by the small visible demand and the fear that any large production will glut the market. Owing to these conditions, the mining of molybdenum has, as already stated, been confined in the past to small-scale operations on high-grade streaks of molybdenite ore, and the methods of recovery have been limited largely to cobbing and hand picking. The unprecedented demand for steel-hardening metals, occasioned by the European war, is, however, slowly drawing the attention of both prospective producers and consumers to the possibilities in regard to molybdenum. Several mining operations looking toward a considerable production of both molybdenite and wulfenite have already been commenced, and, on the other hand, manufacturers are investigating the possibilities of using molybdenum in a large way, and are beginning to realize that it is not as they generally supposed, due to the scarcity of the element in nature and the nonexistence of ore bodies that steady supplies have previously not been available, but rather to lack of development of these ore bodies.

How great a quantity of molybdenum the market will absorb under existing or future conditions is entirely a matter of speculation. It is the author's opinion, however, that the demand for the element will keep pace with a greatly increased production, particularly if the metal is offered at prices below those normally asked for tungsten.

#### HOW MOLYBDENUM CONCENTRATES ARE MARKETING.

The basis on which molybdenum ores and concentrates are bought and sold varies according to whether the contained molybdenum mineral is molybdenite or wulfenite. Molybdenite products are invariably purchased on the basis of their molybdenum content, reckoned as  $\text{MoS}_2$ , whereas wulfenite is always bought either on the basis of its molybdenum content expressed as metallic molybdenum or as  $\text{MoO}_3$ . It is unfortunate that custom should have established



these three methods of calculation where one, based on the content of metallic molybdenum, would have sufficed and avoided needless complications and frequent misunderstandings between buyers and sellers. In connection with the use of these three standards it may be of interest to note that 1 part by weight of  $\text{MoS}_2$  is equivalent to 0.9 part of  $\text{MoO}_3$  and to 0.6 part of Mo, and, inversely, that 1 part by weight of Mo is equivalent to 1.5 parts of  $\text{MoO}_3$  and to 1.67 parts of  $\text{MoS}_2$ .

In the United States the short ton of 2,000 pounds is the measure of weight used in buying and selling molybdenum ore, and quotations are generally based on the number of units of 20 pounds each of pure Mo,  $\text{MoO}_3$ , or  $\text{MoS}_2$  contained in a ton. In Europe, on the other hand, the long ton of 2,240 pounds is almost invariably used, and accordingly European purchasers settle on the basis of a unit containing 22.4 pounds. Quotations both in this country and abroad are generally made on a sliding scale to cover various grades of material. Specifications usually state the minimum percentage of Mo,  $\text{MoS}_2$ , or  $\text{MoO}_3$  in the ore or concentrates that is acceptable to the purchaser, and also the maximum percentage of objectionable elements, such as copper, tungsten, bismuth, arsenic, and antimony that will be allowed. Just what are objectionable elements depends largely on the use to which the molybdenum product derived from the concentrates is destined and the methods employed in treating it. Copper and tungsten seem to be universally objectionable both to the iron and steel and to the chemical trades. Copper is particularly undesirable and its presence in excess of 1 per cent, even in high-grade molybdenite concentrates, is usually sufficient to render the material unmarketable. Some dealers will, however, accept concentrates containing more than 1 per cent copper, but when 4 or 5 per cent copper is present the penalties exacted are such as to be prohibitive. Likewise, some dealers have no objection to the presence of small quantities of arsenic and bismuth in molybdenite concentrates, as they state that these elements are readily volatilized when the ore is roasted in the process of treatment. Up to 1914 it was difficult to sell molybdenite concentrates containing less than 80 per cent  $\text{MoS}_2$ , or wulfenite concentrates with less than 25 per cent  $\text{MoO}_3$ , but at present (March, 1916) it is reported that molybdenite and wulfenite concentrates containing as low as 20 per cent  $\text{MoS}_2$  and 18 per cent  $\text{MoO}_3$ , respectively, can be marketed.

#### PRICES OF MOLYBDENITE AND OF WULFENITE CONCENTRATES.

Under the existing abnormal conditions of the market for all steel-hardening metals it is impossible to quote exact prices for either molybdenite or wulfenite concentrates. The sale of even 1-ton or 2-ton lots is the subject of separate negotiations, and the

prices received vary between wide limits. Based on a content of 90 per cent  $\text{MoS}_2$ , small lots of molybdenite have, during the last two years, brought in this country \$2,500 to \$3,500 per short ton, or \$27.75 to \$38.90 per 20-pound unit of  $\text{MoS}_2$ . The price of wulfenite concentrates during the same period has ranged from \$200 to \$300 per short ton, or from \$10 to \$15 per unit of  $\text{MoO}_3$ , based on a content of 20 per cent  $\text{MoO}_3$ . Of course, the price paid per unit for  $\text{MoS}_2$  or  $\text{MoO}_3$  decreases rapidly with the grade of the material, as is well illustrated by the following schedule of prices quoted in February, 1916, by a domestic manufacturer of molybdenum and ferromolybdenum, for molybdenite concentrates. Based on a content of 50 per cent  $\text{MoS}_2$ , the price offered was \$20 per unit of  $\text{MoS}_2$ , with a penalty or bonus of 20 cents per unit for each per cent below or in excess of 50 per cent  $\text{MoS}_2$ . The prices of various grades of concentrates, according to this schedule, are shown in the accompanying table:

*Prices offered for molybdenite concentrates in February, 1916.*

Per cent $\text{MoS}_2$ in concentrates.	Price offered—	
	Per unit.	Per short ton.
30	\$16	\$480
40	18	720
50	20	1,000
60	22	1,320
70	24	1,680
80	26	2,080
90	28	2,520

As already stated in the discussion of the market for molybdenum, concentrates of both molybdenite and wulfenite must be practically free from deleterious elements, such as copper, tungsten, antimony, arsenic, and bismuth, to command the best prices, and a copper content of more than 2 or 3 per cent generally precludes their sale or else exacts heavy penalties. The following table, which was compiled mainly from quotations furnished through the courtesy of L. Lamy, of Paris, France, shows the fluctuations in the price of high-grade molybdenum concentrates (90 to 95 per cent  $\text{MoS}_2$ ) in European markets since 1907. The table shows that the price has more than trebled in the past three years:

*Prices of high-grade molybdenite concentrates (90 to 95 per cent  $\text{MoS}_2$ ) in European markets, 1908 to 1915.*

Year.	Prices per unit of 20 pounds.	Prices per unit of 22.4 pounds.	Year.	Prices per unit of 20 pounds.	Prices per unit of 22.4 pounds.
		<i>Shillings.</i>			<i>Shillings.</i>
1908.....	\$6.50 to \$7.60	30 to 35	1912.....	\$7.15 to \$11.95	33 to 55
1909.....	5.65 to 7.15	26 to 33	1913.....	10.90 to 18.50	50 to 85
1910.....	6.30 to 6.75	29 to 31	1914.....	19.50 to 31.50	90 to 145
1911.....	6.95 to 9.10	32 to 42	1915.....	23.90 to 36.90	110 to 170

In conclusion, it may be said that prices at present are wholly abnormal, and companies engaged in the mining or concentration of either molybdenite or wulfenite should be prepared for the sharp decline that is almost certain to follow the cessation of hostilities in Europe. Just what the normal price will be is hard to predict, as there are too many uncertain factors to be considered. As already stated, it is the opinion of the author that the demand for molybdenum will continue to increase, particularly if the price of the metal declines so that it closely approaches the normal price of tungsten. Any large production of molybdenum, such as might be derived from the several extensive deposits of low-grade molybdenite ore that are now being developed in this country, would, in all probability, not only cause such a decline in price, but might possibly enable molybdenum to be sold even below the normal price of tungsten. Although it is not safe to make any prediction as to the minimum price at which molybdenum may be marketed in the future, it may be assumed that if the chief use of the metal continues to be in the manufacture of special steels, maximum prices will never greatly exceed twice the corresponding price of tungsten, as the effective ratio of the two metals in steel is slightly above 1:2, and in many instances steel manufacturers could substitute one for the other within certain limits without materially affecting their product.

The present abnormal prices for molybdenite and wulfenite concentrates are, of course, reflected in the quotations on metallic molybdenum and ferromolybdenum. Prices in this country (March, 1916) are nominal at \$4 to \$5 per pound of metallic molybdenum, whereas in England they are slightly lower, averaging about 15s. 6d. or \$3.75 per pound. It is of interest to note that inasmuch as supplies of molybdenum and tungsten are of vital importance in the manufacture of munitions, Great Britain, acting through the governments of her various colonial possessions where ores of these metals are produced, has established a maximum price at which these ores may be sold. As regards molybdenum ore this price is 105s. per unit of  $\text{MoS}_2$ , and for tungsten ore it is 55s. per unit of  $\text{WO}_3$ . Taken in connection with the placing of these metals and their ores and alloys on the list of absolute contraband, this fixing of prices by the British Government has resulted in an extraordinary difference in the relative prices of the two metals in the United States and Great Britain. Quotations on ferrotungsten (75 to 85 per cent tungsten) are (March, 1916) about \$8.50 per pound of contained tungsten in this country as compared with 6s. 6d. to 7s. (\$1.58 to \$1.70) in Great Britain. In other words, molybdenum is being sold in the United States at less than half the price of tungsten, whereas in Great Britain the reverse is the case. However, this great difference in price is not to be wholly accounted for by the conditions mentioned above,



but it is partly due to the fact that the use of molybdenum is much more popular abroad than in this country. For example, some well-known European metallurgists are understood to prefer to purchase molybdenite ore at 120s. per unit of  $\text{MoS}_2$  rather than tungsten ore at 55s. per unit of  $\text{WO}_3$ . In this country, on the other hand, the demand for tungsten is far in excess of that for the comparatively little-known molybdenum.

### QUALITATIVE TESTS FOR MOLYBDENUM.

One of the best and most easily applied qualitative tests for molybdenum is made by treating the finely powdered ore with about 5 c. c. of concentrated nitric acid, evaporating to dryness, treating the residue with  $\frac{1}{2}$  c. c. of concentrated sulphuric acid and again evaporating to dryness. If molybdenum is present a beautiful ultramarine blue color develops in the residue on standing. This characteristic color generally appears in two hours or less, but sometimes 12 hours may be required, depending on the temperature and the humidity.

Another test that is highly recommended is to fuse the finely powdered ore with a mixture of sodium and potassium carbonates (1 to 1) and a small amount of potassium nitrate. The fusion is powdered and extracted with hot water and filtered, the filtrate acidified with hydrochloric acid, the carbon dioxide expelled by boiling, and potassium or ammonium thiocyanate, together with a small piece of metallic zinc, added. If molybdenum is present a bright cherry-red color rapidly develops which disappears on standing in the presence of the zinc. If hydrogen peroxide is added to the solution immediately after the cherry-red color has developed, the color disappears, returning as soon as the peroxide has been reduced. If only small proportions of molybdenum are present, it is suggested that immediately after the thiocyanate and zinc have been added to the acidified filtrate from the leached carbonate fusion, the solution be shaken with a small volume of ether. Any color in the liquid will be extracted by the ether and intensified as a golden brown in the ether ring that forms at the top of the liquid on standing, thus perhaps making visible color that before was imperceptible.

Should there be an imperfect fusion and any extraction of iron in the leach from the fused carbonates, a cherry-red color will immediately develop on the addition of the potassium or ammonium thiocyanate to the solution. With the addition of zinc, however, the color due to iron disappears, and unless a large quantity of iron has been brought into solution the color will completely disappear prior to the development of any color due to molybdenum. If this color reaction of iron and its disappearance on the addition of zinc is borne in mind, the above test for molybdenum is infallible.

Blowpipe and simple chemical tests for the various molybdenum minerals are given on pages 8 to 15.



## QUANTITATIVE DETERMINATION OF MOLYBDENUM.

Previous to the quantitative determinations of the molybdenum in the ores and products obtained therefrom in the concentration tests described in this report, a brief study of published methods for determining the element was conducted by J. C. Morgan, junior chemist, in the Denver office of the Bureau of Mines, to whom the author is indebted for data in the following discussion.

It was found that most of the published methods were open to objections as hereinafter outlined. Further, several precipitation methods that would, without doubt, give correct results on practically pure salts of molybdenum were absolutely inaccurate when applied to certain ores and particularly to low-grade ores. In view of the difficulties encountered, a brief preliminary discussion of the more common methods for the quantitative determination of molybdenum may be of interest here.

## SOLUTION.

The general methods of getting the molybdenum in an ore into solution are as follows: (1) By fusion with a mixture of sodium and potassium carbonates and subsequent leaching of the molybdate formed with hot water; (2) by fusion with a mixture of sodium carbonate and sulphur (1 to 1) and the leaching of the sulpho-molybdate formed with hot water; (3) by fusion with sodium peroxide or sodium hydroxide followed by leaching the molybdate formed with hot water;<sup>a</sup> (4) by solution obtained through evaporation, first with fuming nitric acid, and then with concentrated sulphuric acid.<sup>b</sup>

The fusion methods all have one common disadvantage for ores containing silica in excess of 10 per cent or thereabouts, in that the greater part of the silica is brought into solution and must be removed previous to the precipitation of the molybdenum. Otherwise the molybdenum precipitate is contaminated with silica. The removal of the greater part of the silica necessitates a tedious evaporation, with subsequent filtration and washing of the precipitate. Further, if the alkaline solution is evaporated as in Low's method,<sup>c</sup> the precipitated silica is gelatinous and difficult to wash free from molybdenum salts. If the solution is acidified with hydrochloric acid, as is usual in the precipitation of silica, molybdic trioxide is precipitated on evaporation. One advantage of the fusion methods is that lead, copper, bismuth, iron, and aluminum are left in the residue from the leach, thus precluding the necessity of their removal later.

<sup>a</sup> Darroch, James, and Meiklejohn, C. A., A rapid method of determining molybdenum: Eng. and Min. Jour., vol. 82, 1906, p. 818.

<sup>b</sup> Colett, E., and Eckhardt, M., Bemerkungen zur Bestimmung des Molybdäns im Molybdänglanz: Chem. Ztg., Jahrg. 33, 1909, p. 968.

<sup>c</sup> Low, A. H., Technical methods of ore analysis, 1913, p. 185.

The method of solution by digesting with fuming nitric acid followed by evaporation to dryness, fuming with a small quantity of concentrated sulphuric acid, and diluting, brings all of the molybdenum into solution, if the final residue is washed with dilute ammonia. Any silica or lead present is left in the residue. In contrast to results with fusion methods, copper, bismuth, iron, and aluminum are brought into solution. The removal of these elements, however, seems to offer less difficulty than the removal of silica in the fusion methods.

### PRECIPITATION.

Many methods have been advanced for the precipitation of molybdenum from solution. The chief of these are as follows: (1) Precipitation as molybdenum trisulphide, (2) precipitation as mercurous molybdate, (3) precipitation as lead molybdate, and (4) precipitation as barium molybdate.

The complete precipitation of molybdenum as molybdenum trisulphide may be accomplished in several ways, as follows: (1) By saturating a cold solution, slightly acid with hydrochloric or sulphuric acid, with hydrogen sulphide and heating in a pressure flask;<sup>a</sup> (2) by saturating an alkaline solution with hydrogen sulphide, acidifying with either hydrochloric or sulphuric acids, and heating in a pressure flask;<sup>b</sup> (3) by saturating an ammonical solution with hydrogen sulphide, acidifying with hydrochloric acid, coagulating, filtering, evaporating the filtrate to dryness, expelling the greater part of the ammonium salts, extracting the residue with water containing a small amount of ammonia, adding ammonium sulphide, and acidifying with hydrochloric acid, thus removing the last traces of molybdenum;<sup>c</sup> (4) by adding thioacetic acid and heating the precipitated molybdenum sulphide in a pressure flask.

The precipitation of mercurous, lead, or barium molybdate is accomplished by adding a solution of a salt of these metals to the molybdate solution. With sulphide ores the precipitation of either lead or barium molybdate is impracticable, as no matter what method of solution is used there is invariably enough sulphate present to cause a considerable error through the precipitation of lead or barium sulphate. The prevention of the precipitation of lead sulphate by the use of either sodium or ammonium acetate in hot solution when precipitating lead molybdate is not advisable, owing to the large quantity of acetate required.

The precipitation of mercurous molybdate is usually accomplished by adding a slightly acid mercurous nitrate solution to the solution obtained from the carbonate fusion, to which nitric acid has been

<sup>a</sup> Treadwell, F. P., *Analytical chemistry*, vol. 2, 1912, pp. 285-286.

<sup>b</sup> Treadwell, F. P., *loc. cit.*

<sup>c</sup> Colett, E., and Eckhardt, M., *loc. cit.*

added until the solution is only slightly alkaline.<sup>a</sup> Considerable trouble is experienced in obtaining the right degree of alkalinity. Care must be taken not to obtain an acid solution, as molybdic trioxide may precipitate.

#### CONVERSION OF PRECIPITATE TO WEIGHABLE FORM.

Molybdenum may be weighed as  $\text{MoO}_3$ ,  $\text{MoS}_2$ ,  $\text{PbMoO}_4$ , or  $\text{BaMoO}_4$ . When precipitated as  $\text{MoS}_3$  there are two courses open—either ignition to  $\text{MoO}_3$  or to  $\text{MoS}_2$  in a stream of hydrogen, according to the Rose method. The fact that  $\text{MoO}_3$  is readily volatile at temperatures higher than  $400^\circ$  to  $450^\circ$  C. necessitates great care in igniting. The complete conversion of the  $\text{MoS}_3$  to  $\text{MoO}_3$  requires long ignition at a temperature that should not greatly exceed  $350^\circ$  C. The regulation of this temperature by the ordinary Bunsen flame is extremely difficult and requires much practice. In reducing to  $\text{MoS}_2$  many ignitions are sometimes required to obtain a constant weight.

The conversion of mercurous molybdate to molybdic trioxide by volatilizing the mercury requires extreme care and much time, and it is almost impossible to expel all the mercury without volatilizing some of the molybdic oxide. Lead and barium molybdate precipitates are weighed as such after the precipitates have been dried.

#### VOLUMETRIC DETERMINATION OF PRECIPITATED MOLYBDENUM TRISULPHIDE.

Several volumetric methods have been offered for the determination of molybdenum, but all require that the molybdenum first be freed from the other metals. These methods are, therefore, only of value when applied to ore analysis to determine the molybdenum from a re-solution of the final sulphide precipitate. A valuable volumetric method for such a determination is that of Randall.<sup>b</sup> In this method a solution of the sulphide, acidified with sulphuric acid, is passed through a zinc reductor into a ferric iron solution, and the ferrous iron formed is titrated with permanganate. The advantage that volumetric methods have over gravimetric methods is that they eliminate tedious and difficult ignitions.

In general, practically all molybdenum ores carry silica, together with such metals as iron, aluminum, copper, and lead, so that the most desirable method of analysis would be one in which these elements could be eliminated most expeditiously and completely and give a final molybdenum precipitate readily convertible to a weighable form. A well-tested method that does this most efficiently is that of Colett and Eckhardt.<sup>c</sup>

<sup>a</sup> Low, A. H., loc. cit.

<sup>b</sup> Randall, D. L., The behavior of molybdic acid in the zinc reductor: *Am. Jour. Sci.*, ser. 4, vol. 24, 1907, p. 313.

<sup>c</sup> Colett, E., and Eckhardt, M., loc. cit.



**METHOD ADOPTED BY THE BUREAU OF MINES.**

Inasmuch as the ores treated carried only traces of arsenic or antimony and no tungsten, no account was taken of these metals in the procedure employed. Special precautions are necessary when these metals are present in appreciable quantity. The procedure for the separation of molybdenum from these metals may be found in any of the more complete texts on molybdenum analyses.

Directions covering the method adopted by the Bureau of Mines follow:

Digest the sample of ore—from 0.2 gram to 5 grams, depending upon its seeming richness—with 25 to 35 c. c. of fuming nitric acid in an Erlenmeyer flask for three hours and finally evaporate to dryness. Add 3 c. c. of concentrated sulphuric acid to the residue and heat until dense white fumes are given off in quantity. Cool, dilute to 100 c. c., and filter. Wash the residue with water, allowing the wash water to run into the filtrate. Wash the residue well with dilute ammonia (1 to 3), and then with water. Make the filtrate alkaline with ammonia to precipitate the aluminum and any iron present in the original mineral. Heat, filter, and wash well with hot water. Saturate this alkaline filtrate with hydrogen sulphide to a bright cherry-red color. Filter and wash with hot water. Acidify the filtrate with hydrochloric acid until slightly acid and digest until the precipitated sulphide and sulphur are well coagulated and the excess hydrogen sulphide expelled. Filter on a weighed Gooch crucible. Evaporate the filtrate to dryness in a casserole and drive off the ammonium salts at the lowest possible temperature, being careful not to heat the casserole to redness at any time. Take up the final residue with about 100 c. c. of water to which 5 c. c. of ammonia has been added. Add 10 c. c. of ammonium sulphide, make faintly acid with hydrochloric acid, and digest until the sulphide is coagulated. Filter this on the Gooch crucible used for the previous sulphide filtration. Add an amount of sulphur to the combined sulphides equal to about one-half their weight and ignite over a Bunsen burner at a dull-red heat in a stream of arsenic-free hydrogen for 10 minutes. The ignition may be accomplished by using a Rose crucible cover and tube over the Gooch crucible. Weigh and repeat the ignition as before, until check weights are obtained. The weight obtained is the weight of molybdenum disulphide.



## PART II. DESCRIPTION OF DEPOSITS.

### GENERAL DISTRIBUTION OF DEPOSITS.

A brief field investigation confined to the examination of the better known occurrences of molybdenum in six of the Western and Pacific Coast States, namely, Arizona, California, Colorado, Montana, New Mexico, and Washington, has been sufficient to show that, contrary to general belief, there are in this country large low-grade deposits of both molybdenite and wulfenite ores from which considerable tonnages of marketable concentrates might be obtained.

Examinations of the various deposits were made solely with the purpose of determining their commercial possibilities. Most of those visited were in the prospect stage, and as many of those in the States of Colorado, Montana, and Washington were seen in the late fall when they were covered with snow, some of the following descriptions may seem to appear brief and incomplete. It is hoped, however, that the descriptions and the tabulated information regarding the known occurrences of molybdenum ores in the various States will be of value to those who are interested in deposits of this metal.

### ARIZONA.

Arizona has many deposits of molybdenite and of wulfenite, and with the exception of Nevada it is the only State in which any noteworthy deposits of the latter mineral have been reported. Wulfenite ores from Arizona have supplied the larger part of the molybdenum produced in this country to 1915, and on account of the ease with which they are concentrated they should prove a strong competitor of molybdenite for markets that do not require concentrates with a high molybdenum content.

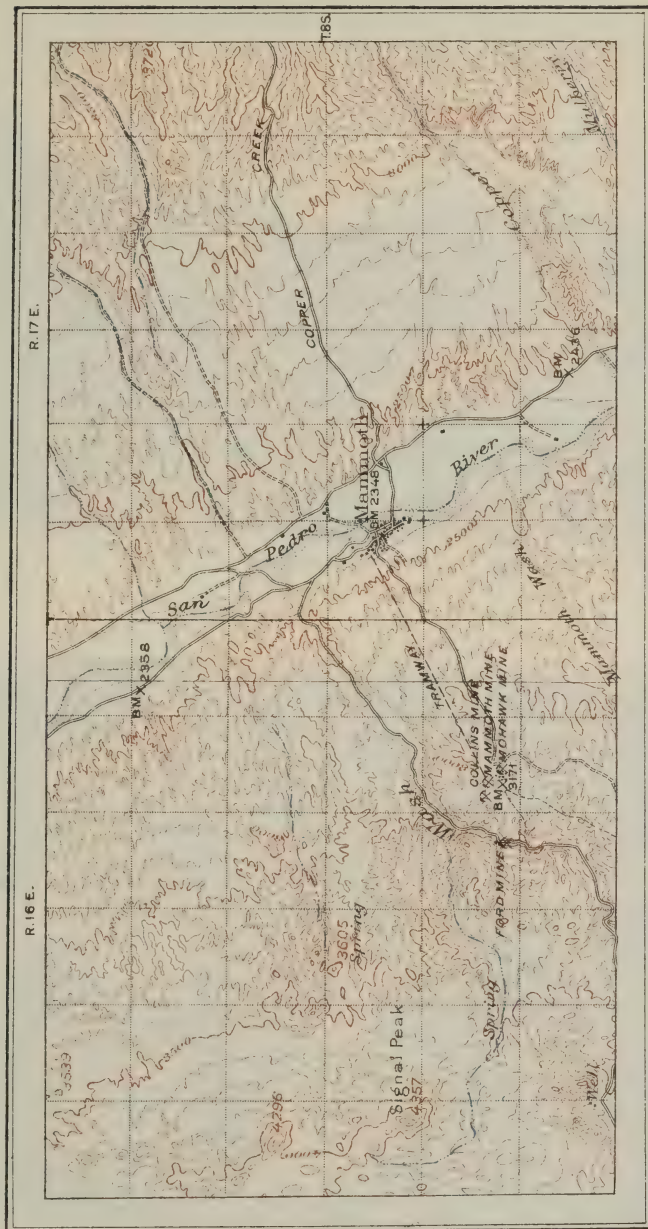
The deposits of wulfenite ore are largely confined to the four southern counties, Cochise, Pima, Pinal, and Yuma, with a few in Gila, Maricopa, and Yavapai Counties. Molybdenite has been reported from only six counties—Greenlee, Gila, Mohave, Pima, Pinal, and Santa Cruz. In most of the Arizona deposits the molybdenite is associated with chalcopyrite and other copper minerals, and inasmuch as the copper in the concentrates must be separated by further treatment the deposits are not as desirable as those free from copper. However, notwithstanding the unfortunate association of copper, some of these deposits will doubtless prove of commercial importance.

### WULFENITE AT THE MAMMOTH AND COLLINS MINES, PINAL COUNTY.

The Mammoth and the Collins are adjoining mines situated at Schultz, in the Old Hat mining district, Pinal County, about 3 miles west of Mammoth and 48 miles north-northeast from Tucson, the railroad point from which supplies are brought over a good wagon road via Oracle. The mines are in section 26, T. 8 S., R. 16 E., in hilly, desert country, on the easterly slope of the drainage basin of the San Pedro River. Their location with reference to Mammoth and the main topographic features of the vicinity are shown in Plate VI. A view of the buildings and dump of the Mammoth mine is given in Plate VII, A. No wood for fuel or for mine timbers is available near by. Fuel oil and distillate for power and wood for timbering are hauled from Tucson. The elevation at the collar of the Mammoth shaft is 3,213 feet, and the collar of Collins shaft about 750 feet to the southwest is 65 feet higher.

The mines are situated on two roughly parallel veins that are about 600 to 700 feet apart. The veins strike in a general northwest to southeast direction and dip to the southwest. The dip of the Mammoth vein varies from  $45^{\circ}$  to  $72^{\circ}$ . Its average dip between the 300-foot and the 750-foot levels is  $69^{\circ} 31'$ , but at the 750-foot level it is only about  $45^{\circ}$ . From the surface to the 300-foot level the average dip of the Collins vein is about  $72^{\circ}$ , but grows flatter below. The veins vary greatly in width, both along the strike and the dip. The Mammoth vein is 1 or 2 inches to 60 feet wide, the part that has been explored south of the main shaft and that roughly constitutes "the south ore shoot" averaging 13 to 15 feet wide for a horizontal distance of about 750 feet. Work on the Collins vein indicates that it has an average width of 12 to 13 feet.

The filling of the Mammoth vein seems to be a mixture of brecciated rhyolite and granite, the whole so highly altered and silicified that its exact classification is impossible. The ore contains wulfenite, vanadinite, descloizite, cerussite, anglesite, chrysocolla, azurite, malachite and small quantities of gold and silver. A few pockets of galena carrying a little silver are found in some of the stopes. The filling of the vein in the Collins mine is similar to that of the Mammoth vein except that it consists almost entirely of altered rhyolite(?). The accompanying minerals in the two veins seem to be identical. The geologic history of the veins is complex and has not been thoroughly studied. Speaking in a general way, however, one may say that they were formed by intrusions of rhyolite(?) along faults in the granitic country rock, the whole being subsequently shattered by faulting and mineralized. The character of the rock of the walls is very irregular; in some places the hanging wall is rhyolite and the

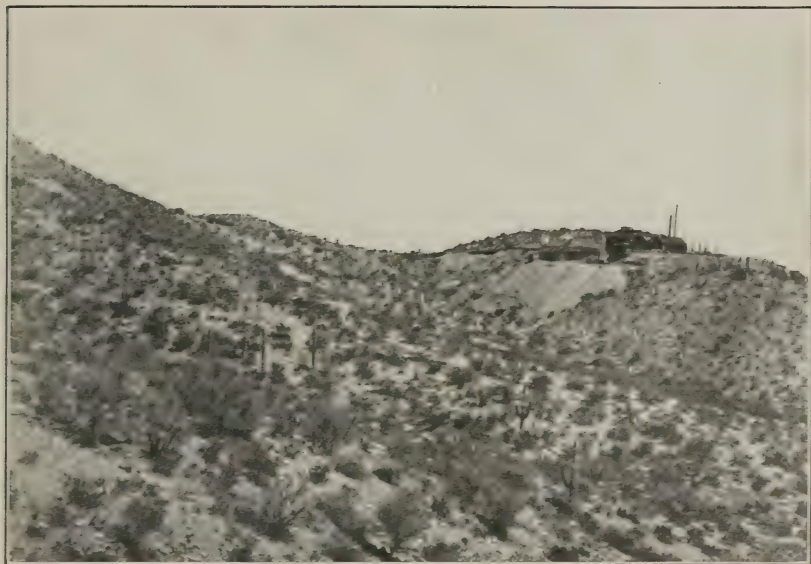


MAP OF THE VICINITY OF MAMMOTH, ARIZONA,  
SHOWING LOCATION OF MAMMOTH MINE

Scale  $\frac{1}{125000}$  1 2 3 4 5 Miles  
Contour interval 100 feet.







A. BUILDINGS AND DUMPS OF MAMMOTH AND COLLINS MINES, SCHULTZ, ARIZ.



B. CRYSTALLIZED WULFENITE (BLACK) IN LEAD CARBONATE (WHITE) FROM MAMMOTH MINE, SCHULTZ, ARIZ. (NATURAL SIZE).

LIBRARY  
OF THE  
UNIVERSITY OF ILLINOIS

foot wall granite, and vice versa. Seemingly the zone of mineralization repeatedly crossed and recrossed the shattered rhyolite. The primary deposition of the economic minerals was probably as sulphides, and the existing minerals may be considered as almost wholly of secondary origin.

Wulfenite is of common occurrence throughout the veins and much of it is beautifully crystallized. The Mammoth mine alone has probably furnished more fine cabinet specimens of the crystallized mineral than any other locality. Most of the crystals are light orange, but they range through various shades of orange, brown, and green, to almost black. Plate III, *B*, shows a remarkably fine group of crystals from this mine. In many places there are 1-inch and 2-inch streaks of nearly pure wulfenite near the center of the ore bodies, and many cracks and fractures throughout the vein material are filled with the mineral. On the 750-foot level of the Mammoth mine the writer noted a shoot of wulfenite and cerussite ore, exposed for 100 feet or more, that averaged 10 to 12 inches wide, and probably contained 20 to 30 per cent of wulfenite. Plate VII, *B*, shows a characteristic occurrence of a veinlet of crystallized wulfenite in cerussite. In many cases the wulfenite is intimately associated with vanadinite, whereas in others the two minerals occur separately.

Veinlets and minute stringers of wulfenite are visible throughout the stopes and in the roofs and floors of most of the drifts. The writer estimates that as a whole the ore bodies contain perhaps 1 to 2 per cent of the mineral. The mines have, however, never been worked for wulfenite, but have been developed chiefly for gold, of which their ores are said to carry \$6 to \$7 per ton. Years ago, 225,000 to 250,000 tons of ore were taken from the Mammoth mine and treated by stamp milling and amalgamation at Mammoth, 3 miles away. Later the old tailing piles were treated for gold by the cyanide process, and still later parts of them were sluiced to extract the wulfenite. At present these tailings are being re-treated for the third time for their wulfenite content, as described on pages 113 to 115.

Several years ago control of both the Collins and the Mammoth mines, which had been idle for many years, was obtained by the Messrs. Young, who installed new machinery and other equipment and did considerable development work with a view of active mining.

Development on the Mammoth mine consists of a main shaft 833 feet deep and about a mile of drifts on eight different levels. The workings on the north side of the shaft have caved badly to within 125 feet of the shaft and are mostly inaccessible. On the 750-foot level it is, however, possible to go 350 feet north of the shaft, but only with great danger. The vein at this level on the north side varies from 6 to 60 feet wide. On the south side of the shaft the principal work has been done on the first, third, fourth, fifth, sixth,

and seventh levels, and aggregates perhaps 4,000 feet of drifts. Much ore has been stoped, but the ground has not caved to any extent. The water level in the Mammoth mine is about 760 feet below the collar of the shaft.

The Collins mine is opened by a tunnel 550 feet long and by a shaft down 700 feet. The principal drifts are as follows: One 325 feet long on the 100-foot level, and one 300 feet long on the 200-foot level. In all, the Collins mine has perhaps 2,000 feet of tunnel and drifts.

The Mammoth mine is equipped with a shaft house, engine and compressor house, boiler house, machine and blacksmith shops, oil storage tanks, etc. Power is generated by two Fairbanks-Morse engines which use "tops" or a No. 2 distillate costing (June, 1915) about 5 cents per gallon in Tucson. One of these engines, rated at 100 horsepower, is belted to a Sullivan angle-compound compressor capable of delivering 528 cubic feet of air per minute at 100 pounds pressure. The other engine, rated at 150 horsepower, is direct connected to a 440-volt generator which supplies power for the mine pump, electric lights, and various small motors used in the machine shop, etc. The hoisting equipment consists of a 12-inch by 14-inch double-drum hoist, which can be operated either by steam or compressed air. The mine pump is situated on the 750-foot level. It is a Prescott direct-connected motor-driven pump with a lifting capacity of 400 gallons per minute against a head of 900 feet.

Both the Mammoth and the Collins mines could, without doubt, produce a large tonnage of low-grade wulfenite ore containing perhaps 2 to 3 per cent of the mineral, and with careful selective mining could produce considerable high-grade ore. It is the writer's opinion, however, that a high-grade wulfenite concentrate could not be produced by treating the ore as a whole by ordinary wet methods on account of its containing considerable cerussite. In fact, a concentrate obtained in this way from ore of ordinary grade from the Mammoth mine showed on analysis only 14.45 per cent molybdic trioxide (corresponding to 39.4 per cent of wulfenite), but 54.54 per cent lead. The vanadinite and copper minerals present in the ore would also tend to detract from the value of the product. However, concentrates of the grade indicated above are salable, and ways might be devised for separating the bulk of the cerussite. It is reported that the wulfenite from these mines contains a small amount of gold.

#### **WULFENITE AT THE OLD YUMA MINE, PIMA COUNTY.**

The Old Yuma mine is situated about 14 miles northwest of Tucson, and 5 miles southeast of the Southern Pacific Railroad in the foothills of the Tucson mountains. To reach it one follows an excellent county road for a distance of 11 miles and then takes a poor road for about 3 miles. The mine is in desert country, and neither water nor timber



is available in the immediate neighborhood. At the mine an inclined shaft had been sunk to a depth of about 300 feet (November, 1913) on the vein, which dips about  $45^{\circ}$  southeast. Other development work consisted of about 300 to 400 feet of drifts, a few feet of raises, and a second shaft which had been abandoned but which was said to be about 100 feet deep.

The hanging wall of the vein is in many places fairly well defined, but inasmuch as the foot wall is indistinct and the crosscuts were few, the writer's estimate of 8 to 10 feet as the average width of the deposit is only approximate. The maximum width observed was about 20 feet in a crosscut on the 65-foot level. The vein material consists of what is seemingly an acid eruptive rock, badly shattered and which has been so highly altered that it is difficult to classify. It carries wulfenite, vanadinite, and cerussite, and in many places is heavily stained with iron oxides.

The wulfenite occurs in seams and cracks throughout the ore in thick, tabular crystals, which are commonly one-eighth of an inch and sometimes even an inch in maximum diameter; also as minute specks and irregular masses in the cavities of the rather porous vein material. The wulfenite crystals generally have a bright orange-yellow color, but some are greenish-black to black. Others are bicolored, having black centers and orange-yellow edges, and in some crystals one entire side may be black and the other side orange. The black coloration of these crystals is of economic interest, as it is seemingly due to the presence of finely divided metallic gold, which can be plainly seen in some crystals with the aid of a high-power microscope. The occurrence of native gold in these crystals is discussed in detail on page 115. It suffices here to say that assays made by the author of the orange-colored crystals failed to show even traces of gold, but that those of hand-picked lots of black crystals showed about 140 ounces of gold per ton.

Much of the wulfenite is associated with either cerussite or well-crystallized vanadinite or both; again, these three minerals may occur entirely distinct from one another.

The quantity of wulfenite present is greatly in excess of the vanadinite and cerussite, but no figures as to the average content of the ore in the three different minerals are available, as to the writer's knowledge the mine has never been sampled with the view of determining such data. A 120-pound lot of ore sent to the Bureau of Mines for concentration tests, which was said to be a sample across a 20-foot face on the 65-foot level of the mine, contained 6.08 per cent molybdic trioxide, which corresponds to about 15.5 per cent wulfenite. This sample was undoubtedly much richer in molybdenum than the average run of ore in the mine, which the writer would estimate as probably containing not over 2 or 3 per cent wulfenite.

It is reported that this mine was originally opened for gold, and that on an average the ore contains about one-half an ounce of this metal per ton. Owing to the distance of the property from a water supply sufficient to run a mill, the mine has never been operated to any extent, and at the time of the writer's visit the only ore that had been extracted was that encountered in development. At that time the only equipment at the property was a small dismantled hoist. No pumping had ever been necessary, and the mine was perfectly dry throughout, even at the bottom of the 300-foot shaft.

The wulfenite in the ore is readily concentrated both by wet and dry methods. If, as demonstrated by the concentration tests of the 120-pound sample of ore referred to above most of the gold is contained in the wulfenite, these concentrates should be of high value, as the molybdenum, lead, and gold contained in them can be readily separated and recovered. For a detailed description of the concentration tests made on ore from this mine, see pages 111 to 115.

The writer has been advised that in 1915 this mine was bonded to Col. Epes Randolph and associates, of Tucson, Ariz., who have erected a mill for concentrating the ore and commenced extensive development of the mine.

#### OTHER OCCURRENCES OF WULFENITE IN ARIZONA.

Besides the wulfenite deposits of the Mammoth, Collins, and Old Yuma mines described above there are many other promising occurrences, particularly throughout the southern part of the State. The Red Cloud, Melissa, and Hamburg mines, which are situated in the Silver mining district, Yuma County, about 80 miles up the Colorado River from Yuma, the nearest point on the railroad, are said to contain a noteworthy quantity of wulfenite in their ores, and have produced many finely crystallized specimens of the mineral. However, these mines have not been operated for many years and have never made any commercial production of wulfenite. Specimens from the Red Cloud and the Hamburg mines show that the wulfenite is often associated with vanadinite. Wulfenite is also reported as occurring at Dome, in the same county, with cerussite in a fluorite gangue, likewise in considerable quantities in several mines in the Castle Dome district, about 30 miles to the north.

In Pima County a property consisting of three and one-half claims owned by Louis Ezekiels, of Tucson, and situated about 25 miles west of that city, is said to show 2 or 3 per cent of wulfenite associated with vanadinite in a well-defined quartz vein. R. O. Boykin, of Tucson, and associates are the owners of a prospect 20 miles from that city and within one-quarter mile of a good wagon road; it is said to show 3 feet of ore carrying 3 per cent of wulfenite associated with vanadi-

nite. Concentration tests are reported to have yielded a product containing about 25 per cent molybdic trioxide. Sufficient water supplies to run a small mill the year round is said to be obtainable from a stream nearby.

According to Pratt<sup>a</sup> there is an occurrence of wulfenite in a mine opened for copper at Troy, in the eastern part of Pinal County, within a few miles of the Gila County line. The wulfenite is said to occur in veins of quartz, which carry  $1\frac{1}{2}$  to 3 per cent of the mineral. Pratt further states that, "A concentrating mill has been erected for separating the wulfenite from its gangue, and the Troy-Manhattan Copper Co., which owns the property, is installing a plant for treating the wulfenite concentrates." Inquiries by the writer failed to show that this property had ever become a commercial producer of wulfenite.

J. Jay Sullivan, of Phoenix, Ariz., has recently submitted to the Bureau of Mines a sample of high-grade wulfenite concentrates in which that mineral is associated with considerable vanadinite. The property from which these concentrates were derived is said to be situated 4 miles from Kelvin, in Pima County.

The Mohawk mine, which is only a few hundred yards from the Mammoth mine at Schultz (Pl. VI), is said to show considerable quantities of wulfenite in its workings. At the time of the writer's visit the property had not been operated for many years, and the workings could not be examined, as they were not in condition to risk entry. In the shaft house there were a few tons of wulfenite concentrates which had been recovered from the dump by leasers. The writer would estimate that these concentrates contained 15 to 20 per cent molybdic trioxide. The wulfenite was associated with considerable magnetite.

J. M. Kellog, of Patagonia, Santa Cruz County, owner of the Blue Lead mine 6 miles north of the town, reports that there is a considerable quantity of wulfenite in its ores. The wulfenite is said to be associated with a large amount of pyromorphite.

The Bureau of Mines has received well-crystallized specimens of wulfenite from C. R. Powers, of Gleason, Cochise County. The vein in which this mineral occurs is said to consist of porphyry and to cut a granite formation. The ore shoot, exposed by a shaft 50 feet deep, is reported as being 12 feet wide. The wulfenite is associated with vanadinite and occurs in seams and stringers throughout the ore.

Wulfenite is also said to be a common constituent of the ores of the Tombstone district, Cochise County,<sup>b</sup> and in places it occurs in considerable quantities. In Yavapai County it is reported to occur

<sup>a</sup> Pratt, J. H., Steel and iron hardening metals: Mineral Resources U. S. for 1903, U. S. Geol. Survey, 1904, p. 342.

<sup>b</sup> Austin, W. L., Silver milling in Arizona: Trans. Am. Inst. Min. Eng., vol. 11, 1882-83, p. 105.



in the ores of several prospects in the Bradshaw Mountains;<sup>a</sup> also with pyromorphite at the Accidental mine of the Poland Mining Co., of Prescott. The latter property is situated in Lynx-Creek Valley near Prescott. The mineral is also said to occur at the Vulture mine in Maricopa County.<sup>b</sup>

#### OCCURRENCE OF MOLYBDENITE AT THE LEVIATHAN MINES, MOHAVE COUNTY.

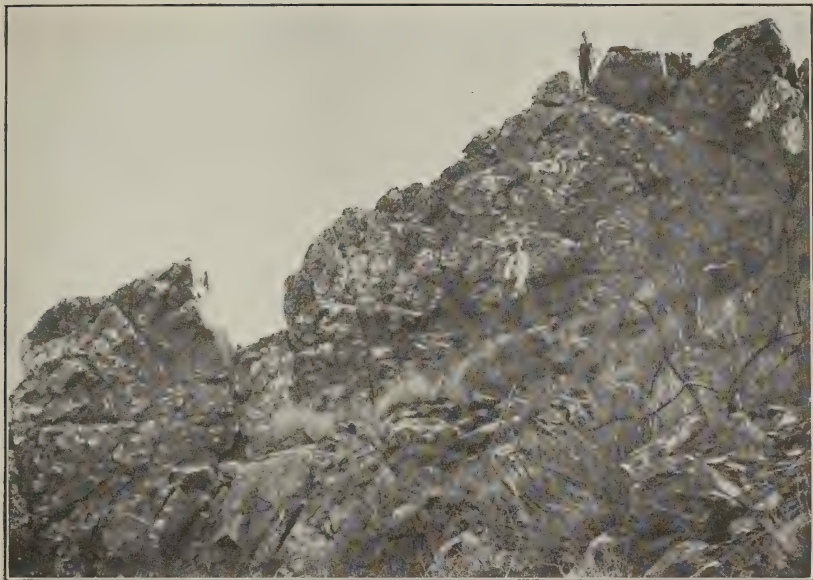
The molybdenite property of the Leviathan Mines Co. is situated in Copper Canyon, Mohave County, on the eastern slope of the Hualpai Mountains. It is in the Cedar Valley mining district, about 3 miles southeast of Copperville and about 25 miles east of Yucca, the nearest point on the Atchison, Topeka & Santa Fe Railway. The property is reached from Yucca by a good wagon road to Copperville, thence by trail down Copper Canyon to the claims, which at their low point in the canyon bottom are at an altitude of about 4,000 feet. The country is arid and as vegetation is confined to desert types no timber is available. Fortunately, water for domestic use can be obtained from springs in the canyon and additional supplies for other purposes from the mine seepage. The wagon road from Yucca to Copperville passes over the main range of the Hualpai Mountains, and reaches an elevation of about a mile. The altitude at Yucca is 1,789 feet, and from this point to the base of the mountains on the west the road has only a slight grade, averaging perhaps 1 or 2 per cent, but on the Yucca side of the range proper the grade is about 7 per cent, and on the Copperville side there are grades as high as 11 per cent. Notwithstanding these steep pitches the road is fairly passable, and auto trucks with loads of 4 to 5 tons make the trip from Copperville to Yucca in about two hours.

The Leviathan properties consist of a group of six claims located on two approximately parallel veins known as the "Whale" and the "Copper Wonder." These veins traverse a granite country rock and consist of white quartz carrying molybdenite and chalcopyrite as the principal ore-forming minerals. The strike of both veins is approximately north-south, and they dip between 85° and 90° west. As the granite country rock is more easily weathered than the hard quartz vein material, erosion has left the veins sticking into the air and made their outcrops easily traceable. In some places the outcrop of the Whale vein is 30 to 40 feet high and its outcrop on the south side of the canyon can be seen for miles. Plate VIII shows views of this outcrop, which can be traced for more than 1,500 feet. The Whale vein which is the larger of the two, varies in width from 6 to 40 feet, and the Copper Wonder vein from about 2 to 20 feet.

<sup>a</sup> Sanford, S., and Stone, R. W., *Useful minerals of the United States*: U. S. Geol. Survey Bull. 585, 1914, p. 19.

<sup>b</sup> Dana, E. S., *A system of mineralogy*, 6th ed., 1911, p. 991.

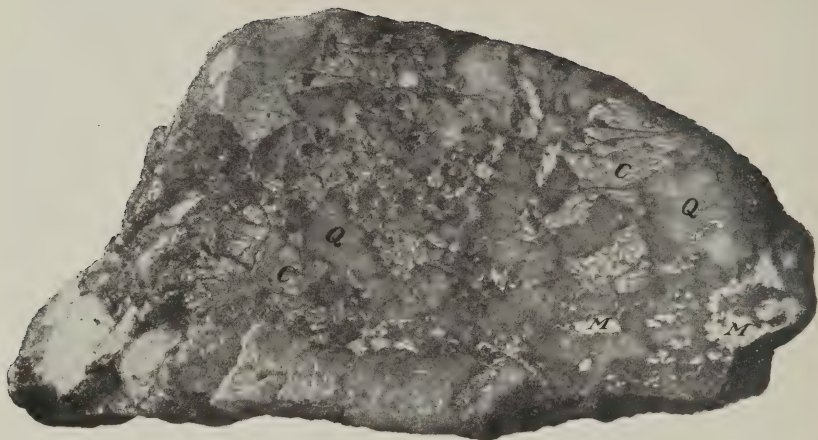




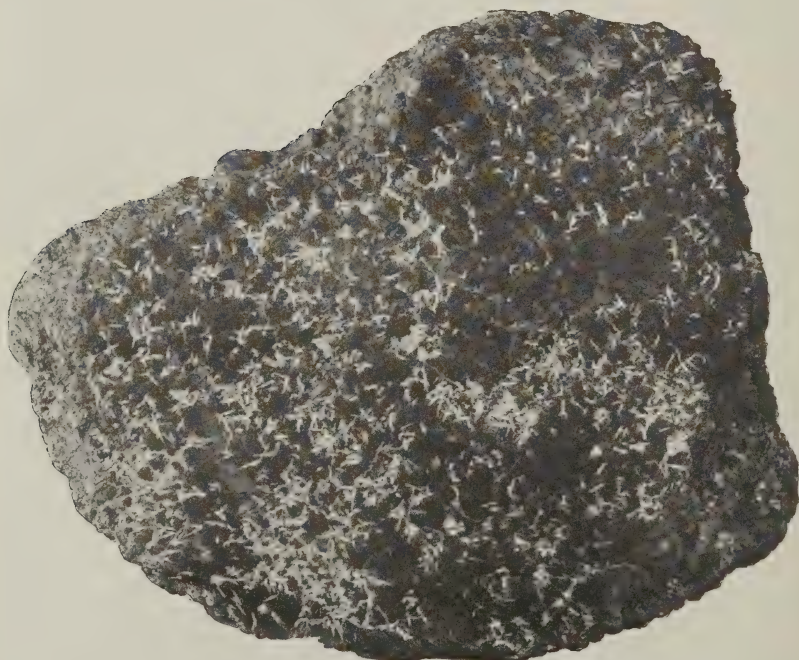
A. NEAR VIEW OF OUTCROP OF WHALE VEIN, COPPER CANYON, NEAR COPPERVILLE, ARIZ., SHOWING ITS HEIGHT.



B. OUTCROP OF WHALE VEIN, LOOKING SOUTH. HEADFRAME OF SHAFT AND PART OF WASTE DUMP ARE SHOWN IN MIDDLE FOREGROUND.



A. SECTION THROUGH TYPICAL ORE FROM WHALE VEIN, COPPER CANYON, NEAR COPPERVILLE, ARIZ. (TWO-THIRDS NATURAL SIZE.) M, MOLYBDENITE; C, CHALCOPYRITE; Q, QUARTZ.



B. SECTION OF TYPICAL MOLYBDENITE ORE FROM LEADER MINE, HELVETIA, ARIZ. (ONE AND ONE-HALF TIMES NATURAL SIZE.)



The molybdenite occurs in amorphous and finely crystalline form in thin veinlets and irregular masses throughout the quartz, and as a fine crystalline powder and as nuggets in vugs and cavities in the veins. Much of it is somewhat intimately associated with chalcopyrite, and in only a few places could the writer obtain specimens that were free from copper. In some places, the individual masses of molybdenite and chalcopyrite are large and are separated by masses of quartz, and in other places they may be very small and closely intermingled. Plate IX, A, illustrates a section through a piece of typical ore showing the relation of the two minerals. Where weathering has taken place both the molybdenite and the chalcopyrite have formed alteration products, the former changing to the yellow molybdate, and the chalcopyrite to malachite and azurite. Secondary copper sulphides have also been formed and in several places on the Whale vein native copper occurs. In this vein there are also occasional occurrences of pyrite and small patches of sphalerite, and the ore is reported to carry about 0.02 ounce of gold and 1.4 ounces of silver per ton.

The country rock is a medium-grained gray granite consisting of quartz, feldspar (orthoclase, microcline, and plagioclases varying from albite to labradorite) biotite, muscovite, and small amounts of the usual accessory minerals, zircon and apatite. In the specimen examined there was much zonal development in the feldspar and also much alteration to sericite and possibly kaolin. The rock is more monzonitic in character than the usual granite, but it is a granite nevertheless.

Development work has practically been confined to the Whale vein. At the time of the writer's visit to the property in June, 1915, it consisted of two tunnels driven along the strike of the vein only a few feet above the level of the canyon floor on opposite sides of the gulch, and of a shaft sunk on the north side of the canyon. The tunnel on the south side had been driven for about 210 feet, and that on the north for about 30 feet. The shaft was 100 feet deep and a drift had been driven from the bottom in a southerly direction for 20 feet.

The writer estimates that the average grade of ore exposed in the workings and in the outcrop contains between 2 and 3 per cent  $\text{MoS}_2$ , and  $1\frac{1}{2}$  to 2 per cent copper. This estimate is substantiated by the results of 22 analyses, each representing samples taken over widths of 2 to 17 feet at various points in the tunnel and shaft and on the outcrop. These analyses, which were kindly furnished the writer by R. C. Jacobson, general manager and engineer of the company, averaged 2.73 per cent  $\text{MoS}_2$ , and 1.71 per cent copper, and indicate that the percentages of molybdenum (metal) and copper in the ore are approximately equal. Several rich streaks in the ore body are reported as containing 5 to 8 per cent  $\text{MoS}_2$  over widths of 4 to

6 feet. The ore that is richest in molybdenite appears to contain relatively less copper than that of average grade.

No particular difficulty should be encountered in concentrating this ore either by electrostatic or flotation methods, but the concentrates obtained would consist of a mixture of molybdenite and chalcopyrite, probably in almost equal parts, and in that form their molybdenite content would be of little value. A retreatment of the concentrate by any one of a number of mechanical methods would probably separate the greater portion of the copper from the molybdenite (see pp. 94 to 109), but the writer is of the opinion that none of these processes would completely eliminate the copper from a product containing such a large percentage of it. If such is the case, chemical methods of separation must be employed. The results of a concentration test on this ore by the Wood flotation process are given on page 104.

At the time of the writer's visit to the property, surface equipment consisted of a shaft house containing a Western Iron Works 10-horsepower gasoline hoist, a compressor house containing an Alamo 20-horsepower gasoline engine (type S) belted to a Clayton two-drill air compressor, a blacksmith shop, and a cookhouse. The miners were quartered in tents. The little water seeping into the shaft was removed by means of a small Cameron sinking pump, which delivered it when required to a 3,500-gallon galvanized-iron storage tank. This water supply was used for cooling the engine jackets.

It is estimated that the cost of mining will vary from \$1 to \$1.50 per ton, according to the scale on which it is carried on. A water supply sufficient for milling purposes could probably be developed and brought to the property at reasonable cost. Moreover, on account of the location of the mine in a canyon having a considerable drainage area, a considerable quantity of mine water will in all probability have to be pumped with further deepening of the shaft and extension of the workings. This water might be used as a supplementary supply for milling. At present the cost of transportation to and from the railroad at Yucca is high, but if a road were built connecting the claims with the road at Copperville this cost would probably not exceed \$7 or \$8 per ton.

The outcrop of the Whale vein is so exceptionally well defined and so wide and the molybdenite and the copper contents of the vein, as indicated by sampling of the outcrop and workings, are so regular and persistent that the writer considers the property a most promising one notwithstanding the unfortunate association of copper with the molybdenite.



**MOLYBDENITE AT THE SMITH & SAWYER AND THE MILLER CLAIMS, MOHAVE COUNTY.**

Just below the property of the Leviathan Mines Co. in Copper Canyon and within a distance of 2 miles there are 16 or 17 other well-defined veins, all containing molybdenite associated with chalcopyrite in white quartz, striking in approximately the same direction and having the same general characteristics as those of the Whale and the Copper Wonder veins. Nine claims have been located on these deposits, one group of five claims belonging to Smith & Sawyer, of Kingman, and a group of four owned by J. F. Miller, of Copperville. As far as can be determined from an inspection of their outcrops, the average width of these veins is seemingly 1 to 10 feet and their average content in molybdenite and copper is approximately the same as in the Whale vein, that is, 2 to 3 per cent  $\text{MoS}_2$  and  $1\frac{1}{2}$  to 2 per cent copper. However, the relative proportions of molybdenite and copper in the different veins seem to vary considerably, but inasmuch as little development work has been done on any of the deposits, this seeming variation may not be true of the veins as a whole but may be solely due to local differences in the individual veins.

The molybdenite in the outcrops of most of the veins has been extensively altered to molybdite, and one of the veins of the Smith and Sawyer group in particular has furnished many remarkably fine crystallized specimens of this mineral. A considerable quantity of bornite was noted in the ore from a vein on the Miller property.

Inasmuch as the mining, concentration, and transportation problems involved in working these veins are identical with those presented by the deposits of the Leviathan Mines Co., they will not be dealt with here.

Considered as a whole, this system of approximately parallel veins on the Leviathan, Smith & Sawyer, and Miller properties constitutes one of the notable molybdenite deposits of the United States.

**MOLYBDENITE PROPERTY OF ARIZONA MOLYBDENUM CO., PIMA COUNTY.**

The molybdenite property of the Arizona Molybdenum Co. is situated in the northerly end of the Baboquivari Mountains, Pima County, about 35 miles from the railroad at Twin Buttes, the terminal station of a branch line from Sahuarita on the Southern Pacific Railroad. It consists of 10 mining claims of approximately 20 acres each. The ore as represented by a 5-pound sample sent to the writer is seemingly a pegmatite which consists almost entirely of quartz and highly altered feldspar, and throughout which molybdenite, chalcopyrite, and small quantities of galena occur. The ore is reported to contain gold

and silver, and a lot said to be an average sample from a width of 8 feet at one point in the vein is stated to have assayed 2.2 ounces of silver, 1.8 ounces of gold, and 3.9 per cent copper. The vein is reported to be 4 to 10 feet wide and to strike southeast. It is said to traverse a coarse-grained granite. Development on the property consists of three shafts, 36, 40, and 60 feet deep, and of about eight smaller shafts each 10 to 12 feet deep. F. J. Wharton, of Tucson, is the president of the company.

#### MOLYBDENITE AT LEADER MINE, PIMA COUNTY.

The Leader mine, of the Helvetia Copper Co., is situated near the town of Helvetia, Pima County, at an altitude of about 4,700 feet, on the westerly slope of the Santa Rita Mountains, about 27 miles southeast of Tucson. The main tunnel by which the property is opened runs in a general northerly direction, on the contact of an altered limestone with an underlying granite. In operating the mine for copper a number of years ago, an occurrence of molybdenite was encountered at a depth of about 40 feet in a winze sunk from the tunnel level, about 150 feet in from the portal. At the time of the writer's visit to the property, this part of the winze was inaccessible. The following information regarding the occurrence is taken from a description by Schrader and Hill,<sup>a</sup> who visited the mine in 1909.

The winze referred to above is sunk on a quartz vein containing various copper minerals, and dipping at an angle of about 40°. At the tunnel level this vein is only a few inches in width, but there is 1 to 3 feet of mineralized limestone on either side of it, containing chalcopyrite, pyrite, coarse calcite, and quartz, the whole constituting a low-grade copper ore. At a depth of 35 or 40 feet in the winze, molybdenite occurs in both the vein and the surrounding country rock. Eight or ten feet farther down where the dip of the vein flattens the chalcopyrite and pyrite practically disappear, and there is a great increase in the quantity of the molybdenite. At this point there is a body of relatively pure molybdenite ore at least 3 or 4 and perhaps 6 or 7 feet thick. The molybdenite occurs as lenses, irregular bunches, and crystal aggregates associated with both the quartz and the highly altered limestone. Its most common form is as small, flaky crystals.

The rock in which the molybdenite occurs is described as a "dull-brownish and greenish to yellowish mineralized garnetiferous silicified limestone and quartz." It is somewhat massive, and varies from fine to medium grained. It is composed largely of pale-greenish garnet corresponding to grossularite. Quartz is the next most

<sup>a</sup> Schrader, F. C., and Hill, J. M., Some occurrences of molybdenite in the Santa Rita and Patagonia Mountains, Arizona: U. S. Geol. Survey Bull. 430, 1910, pp. 156-157.

abundant mineral, with calcite, magnetite (or ilmenite), and epidote present in small quantities as secondary minerals.

Plate IX, *B*, shows a section through a typical piece of molybdenite ore picked up by the writer from a pile of 2 or 3 tons on the dump at the mine. The molybdenite is seen as small flaky crystals (white) scattered through the groundmass of the highly altered rock.

The ore as represented by the small dump above mentioned contains perhaps 5 to 6 per cent molybdenite. Since the writer's visit to the property some development work has been done on the molybdenite ore and a few tons gotten out. In the annual report of the company for 1915 sales of molybdenite ore during the year are said to have netted \$1,891, and it is stated that there were approximately 100 tons of ore in sight that should net about \$26 a ton.<sup>a</sup>

#### OTHER OCCURRENCES OF MOLYBDENITE.

It is reported that a few tons of rich molybdenite ore was taken out several years ago by the Cuprite Copper Co., of Tucson, in mining a large pocket of copper ore on its property in the northwest end of the Santa Rita Mountains, in Pima County. The property is about 8½ miles south of Vail, the nearest point on the Southern Pacific Railroad, and at an altitude of about 4,000 feet. No molybdenite ore in place could be found by the writer in a hasty examination of the mine workings, but several specimens containing 10 to 20 per cent of the mineral were picked up on the mine dump. In these the molybdenite was associated with chalcopyrite and copper carbonates in a gangue of highly altered and silicified limestone. No evidence as to the form of the molybdenite deposit or the possibilities of further occurrences of the mineral could be obtained.

E. O. Stratton, of Tucson, and associates are the owners of a molybdenite property situated on Marble Creek, in the Santa Catalina Mountains, in Pima County, about 20 miles northwest of Tucson and 15 miles south of Oracle. Some promising showings of molybdenite are said to have been encountered in prospect work.

Other occurrences of molybdenite in Pima and Santa Cruz Counties are as follows:

At the Ridley mine, owned by C. B. Ridley, of Helvetia, and situated in the foothills about 1½ miles southwest of that town.

At the McCleary prospects in Madera Canyon, about 10 miles south-southwest of Helvetia and 35 miles south of Tucson.

In the foothill part of Providencia Canyon, about 10 miles northeast of Nogales and 5 miles north of the United States-Mexican boundary line near the Golden Rose and at the Buena Vista mine, the latter

---

<sup>a</sup> Editorial Eng. and Min. Jour., vol. 101, 1916, p. 482.



owned by the Banco del Oro Mining Co., of Magdalena, Sonora, Mexico.

At Duquesne, about a quarter of a mile west of the Belmont mine, on property owned by Capt. O'Connor, of Duquesne.

At the Benton mine and the Line Boy prospect in San Antonio Canyon, about 2 miles south of Duquesne and 15 miles north-north-east of Nogales. The Benton mine is owned by Dennis Coughlin and partners, and the Line Boy prospect by Capt. O'Connor, all of Duquesne.

In all these localities the molybdenite occurs in granite or in quartz traversing the granite. None of the properties has been developed beyond the prospect stage, and present showings do not indicate that any of them are of commercial importance. The above information concerning these occurrences was obtained from detailed descriptions by Schrader and Hill.<sup>a</sup>

It is stated <sup>b</sup> that one of the mines of the Santa Nina Co., of Patagonia, has a 6-inch or 7-inch vein of high-grade molybdenite ore in one of their copper mines.

Molybdenite is found in the ores of the Clifton-Morenci district, Greenlee County, in fissure veins with pyrite, chalcopyrite, and sphalerite.<sup>c</sup> It also occurs in the disseminated ores at Miami, Gila County, in the primary ores of the chloride district, Mohave County, in a granite-gneiss, and as an original mineral in the ores at the Ray and Kelvin mines in Pinal County.<sup>d</sup>

### CALIFORNIA.

Although there has never been any commercial production of molybdenum ore in California, molybdenite is widely distributed throughout the State and development work on some of the known occurrences of the mineral will probably prove them to be of economic importance.

Following is a list of deposits that have come to the author's attention. The list includes 41 occurrences of molybdenite in 16 different counties. Brief notes regarding six of these occurrences follow. Wulfenite has been reported in a few localities, principally in the southern part of the State, but as far as known to the writer none of the deposits of this mineral is of commercial interest.

---

<sup>a</sup> Schrader, F. C., and Hill, J. M., Some occurrences of molybdenite in the Santa Rita and Patagonia Mountains, Arizona: Contributions to economic geology: U. S. Geol. Survey Bull. 430, 1910, pp. 154-162.

<sup>b</sup> Private communication from J. M. Kellogg, Patagonia, Ariz.

<sup>c</sup> Lindgren, W., The genesis of the copper deposits of Clifton-Montmorenci, Arizona: Trans. Am. Inst. Min. Eng., vol. 34, 1905, pp. 515, 523.

<sup>d</sup> Sanford, S., and Stone, R. W., Useful minerals of the United States: U. S. Geol. Survey Bull. 585, 1914, p. 16.



*Occurrences of molybdenite in California.*

Location.	Remarks.	Reference.
Calaveras County:		
New Big Tree.....	Associated with feldspar.....	Specimen in U. S. National Museum.
Zeila mine.....	Associated with quartz, sericite, and pyrite.	Do.
Eldorado County:		
Fairplay.....	Small crystals with calcite, chalcopryrite, garnet, and epidote.	Specimen in Agassiz Museum.
Near Red Hill, 9 miles east of Sanger.	Flakes up to $\frac{1}{4}$ inch in diameter in white quartz.	Specimens in collection of U. S. Geol. Survey.
36 miles northeast of Millwood.	Fair-sized flakes in white quartz.	Specimen in California Bureau of Mines Museum.
Grizzly Flat.....	Large flakes.....	Do.
Fresno County:		
Kings River Canyon, copper mine.	Flakes in quartz.....	Do.
Green Mountain, south fork of San Joaquin River.	.....	Do.
Fresno Flat.....	.....	Do.
Buchanan prospect, west of Granite Creek, south fork of Kings River.	See p. 62.	
Near Soda Springs.....	In quartz.....	Do.
Inyo County:		
Bishop, near Hillside Dam, south of Bishop Lake.	See p. 61.	
White Mountains.....	.....	Do.
West Arm of Death Valley, south of Lida.	.....	Eng. and Min. Jour., vol. 81, 1906, p. 205.
Kern County:		
Gold region near Havilah.....	.....	U. S. Geol. Survey Bull. 585, 1914, p. 33.
Near Caliente.....	See p. 63.	
2 $\frac{1}{2}$ miles northeast of Randsburg.	Deposit said to be 30 feet wide and traced for 2 miles on limestone contact.	Eng. and Min. Jour., vol. 96, 1913, p. 186.
Los Angeles County:		
Northeast part of county.....	.....	Mineral Resources U. S. for 1901: U. S. Geol. Survey, 1902, p. 266.
Los Angeles.....	Associated with feldspar and quartz.	Specimen in U. S. National Museum.
Mono County:		
12 miles northwest of Bridgeport, on west fork of Walker River.	Molybdenite-bearing core about 20 feet wide in quartz vein associated with molybdenite.	Mineral Resources U. S. for 1901: U. S. Geol. Survey, 1902, pp. 265-266.
Mono Lake.....	Crystals on quartz.....	Specimen in Agassiz Museum.
Cameron, near Bridgeport.....	.....	Specimen in California Bureau of Mines Museum.
Minnie mine, Sweetwater Range.	In quartz.....	Do.
Nevada County:		
Mariposa mine, Rosario district.	.....	Do.
Mayflower mine, Nevada City..	Flakes up to 2 inches in diameter in quartz.	Do.
Placer County:		
Golden Stag mine, Ophir region.	Small flakes.....	U. S. Geol. Survey Bull. 585, 1914, p. 33.
On Red Mountain, 3 miles north of Cisco.	Crystalline molybdenite in white quartz.	Specimens in collection of U. S. Geol. Survey.
Riverside County:		
4 $\frac{1}{2}$ miles northeast of Corona...	See p. 62.	
About 17 miles west of Hemet, in sec. 3, T. 5 S., R. 2 E.	Fair sized flakes in iron-stained quartz.	Do.
16 miles from Corona.....	In siliceous gangue.....	Mineral Industry, vol. 7, 1898, p. 514.
San Bernardino County:		
Leastalk.....	Fine grains in white quartz with chalcopryrite.	Specimens in collection of U. S. Geol. Survey.
San Diego County:		
40 miles east of San Diego.....	See p. 60.	
Near Dulzura.....	Small flakes with quartz and ferro-magnesium minerals.	Specimens in collection of U. S. Geol. Survey.
Campo.....	In granite.....	Specimen in California Bureau of Mines Museum.
Do.....	Associated with feldspar and quartz.	Specimen in U. S. National Museum.
Dewey mine at Grapevine, near Warner's Hot Springs.	.....	Mineral Industry, vol. 11, 1902, p. 477.
Shasta County:		
Tom Neal Mountain, 5 miles from Delta.	In granite.....	Specimen in California Bureau of Mines Museum.
Hazel Creek.....	do.....	Do.
Lamoine.....	.....	U. S. Geol. Survey Bull. 585, 1914, p. 33.

*Occurrences of molybdenite in California—Continued.*

Location.	Remarks.	Reference.
Tulare County: Three Rivers.....	.....	Specimen in California Bureau of Mines Museum.
California Hot Springs.....	See p. 62.	
Tuolumne County: West of Tower Peak.....	In quartz gangue on a contact of marble and a white schistose pyroxene with epidote and sphalerite.	Turner, Am. Jour. Sci., vol. 5, 1893, p. 427.
South of Knight Creek, north-east of Columbia.	In granite.....	Do.
Ventura County: On Alamo Mountain, north-eastern part of county.	Vein said to be 8 to 15 feet wide..	Mineral Resources U. S., 1901, U. S. Geol. Survey, 1902, pp. 265-266; Eng. and Min. Jour., vol. 72, 1901, p. 337.

**MOLYBDENITE AT PROPERTY OF SANTA MARIA MOLYBDENUM MINING & MILLING CO., SAN DIEGO COUNTY.**

The property of the Santa Maria Molybdenum Mining & Milling Co., of San Diego, is situated about 40 miles east of that city in the hills to the southeast of the San Pasquale valley and is readily reached from San Diego by automobile, the road being excellent. It is understood to consist of 160 acres of patented ground.

The molybdenite-bearing deposit consists of a large granite dike varying in width from 35 to 100 feet and striking in a general northeasterly direction. The outcrop of this dike is about half a mile long and is particularly well defined. Plate X, *A*, shows a view looking toward the northwestern end, where the outcrop terminates in an almost perpendicular cliff about 200 feet high. The canyon on which this cliff faces drains into San Pasquale valley which may be seen in the distance in Plate X, *A*. A good idea of the size of the deposit may be gained from a comparison with the size of the persons shown in the illustration.

The molybdenite occurs sparingly throughout the granite in crystal aggregates and irregularly shaped masses of radial structure, most of which vary in size from one-eighth to 1 inch in maximum diameter. It is associated with a little pyrite. Both the molybdenite and the pyrite have been altered, the former slightly and the latter extensively. In a few instances the transformation of the molybdenite to molybdite has been complete, the latter mineral crystallizing in radial groups, such as are shown in Plate V, *B*, which illustrates a specimen of molybdite from this property. More often, however, the molybdenite has been only partly altered, resulting in many interesting specimens, one of which is pictured in Plate II, *B*. The iron oxide formed by the alteration of the pyrite has stained the larger part of the outcropping granite a light red. In this stained part the molybdenite areas are almost invariably surrounded by a narrow rim of unstained rock, as is rather clearly illustrated in Plate

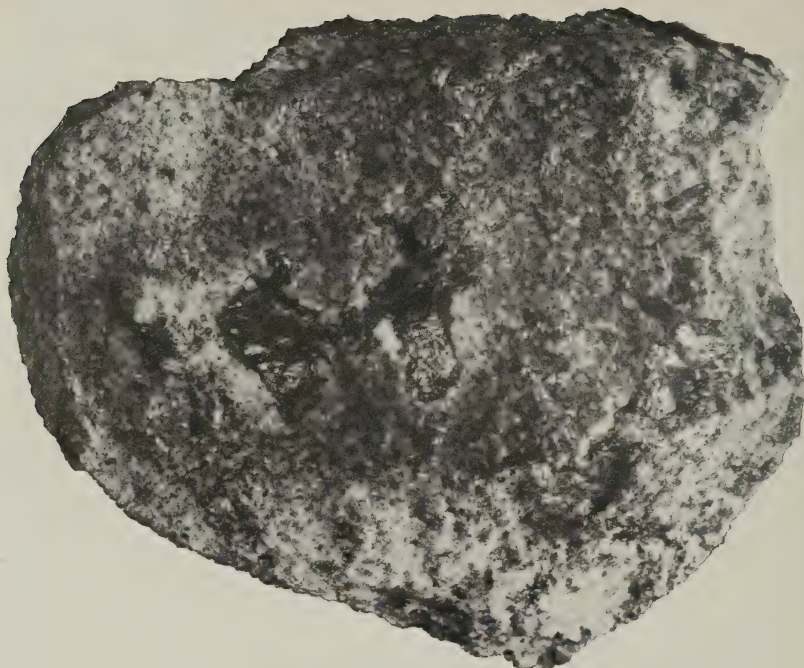


1. OUTCROP OF DIKE ON SANTA MARIA PROPERTY, SAN DIEGO COUNTY, CAL., LOOKING NORTHWEST.

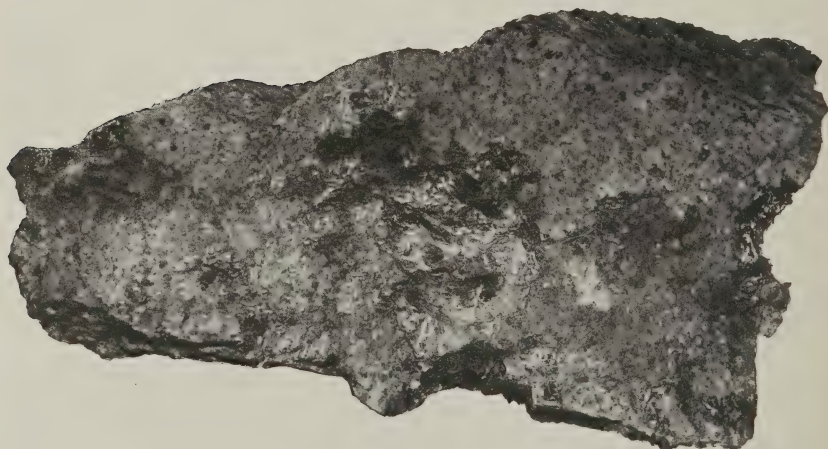


2. OUTCROP OF LOW-GRADE MOLYBDENITE ORE AT PRIMOS MINE, NEAR EMPIRE, COLO.





A. MOLYBDENITE IN IRON-STAINED GRANITE FROM SANTA MARIA PROPERTY, SAN DIEGO COUNTY, CAL. (FOUR-FIFTHS NATURAL SIZE.)



B. MOLYBDENITE IN GRANITE FROM JOHN FLETCHER QUARRY, NEAR CORONA, CAL. (TWO-THIRDS NATURAL SIZE.)



XI, A, which shows a typical specimen of molybdenite ore from the property.

The granite of which the dike is composed is medium grained, and where not iron stained it is light gray to almost white. A microscopical examination of it shows the presence of the following minerals: Quartz, orthoclase, plagioclases varying from albite to oligoclase, biotite, a few minor accessory minerals, and pyrite.

The only development work on the property at the time of the writer's visit in 1914 was five or six open cuts partly crosscutting the dike. Some of them showed ore which might average from 0.25 to 0.5 per cent molybdenite, but from all indications the dike as a whole contains a very much smaller percentage of the mineral. Unless further development work discloses areas richer in molybdenite, the writer does not believe that the occurrence will prove of economic importance. However, as the character of the ore is such that the molybdenite can be readily concentrated into a high-grade product, the discovery of richer ore might lead to successful operations.

A lot of ore weighing 313 pounds, kindly shipped the Bureau of Mines by the owners of the property for concentration tests, and said to be unsorted, assayed 0.47 per cent  $\text{MoS}_2$ . For the results of hand-picking and concentration tests of this ore, see pages 97 to 99.

#### MOLYBDENITE NEAR BISHOP, INYO COUNTY.

Specimens of molybdenite associated with molybdite in white quartz have been submitted to the bureau by M. C. Hall, of Bishop. The deposit from which the ore came is said to be situated at an altitude of about 10,000 feet in the Sierra Nevada Mountains about 20 miles from Laws, the nearest station on the Southern Pacific Railroad. There is a road from Laws passable by automobile to within a mile and a half of the property, thence a poor wagon road for a mile, but no road for the last half mile. It is stated, however, that one could readily be built to cover the remaining distance. The outcrop of the deposit is reported to be about 100 feet long and the same distance across. Practically no development work has been done. In a 40-pound sample said to represent average ore, the author would estimate the molybdenite content to be 1 to 2 per cent. With the exception of the occasional occurrence of small grains of chalcopryite, no minerals other than molybdenite and its alteration product, molybdite, were noted in the quartz. The molybdenite occurs in flakes varying in size from minute specks to an eighth of an inch or more in maximum diameter, and the ore is seemingly of such a nature as to be readily concentrated by either electrostatic or flotation methods. Development work may perhaps prove the deposit to be of economic importance.

**MOLYBDENITE AT THE BUCHANAN PROSPECT, FRESNO COUNTY.**

Specimens of molybdenite ore have been submitted to the bureau by Paul Buchanan, of Kearney Park, Cal. They are said to have come from a deposit situated on the north side of the south fork of Kings River, between Granite and Copper Creeks, and about 2 miles west of Kanawyer's Camp, in T. 13 S., R. 31 E. The property is difficult of access, being 25 miles or more from the nearest wagon road at Hume, which in turn is a greater distance from the nearest railroad station. The vein in which the molybdenite occurs is reported to have an average width of 1 to 3 feet and to be traceable on the surface for about 3,000 feet, but the outcrop carries molybdenum for only about 2,000 feet. The strike of the mineralized part of the vein is said to be almost north and south. The specimens submitted consist of granite cut by small quartz stringers occasionally containing flakes of molybdenite up to one-fourth inch in maximum diameter. The molybdenite is associated with a small quantity of chalcopyrite. The granite consists of quartz, feldspar, and biotite, and is somewhat iron stained. If the average width of the vein is only 1 to 3 feet, as stated, and the grade of the ore is not richer than the samples submitted, the inaccessibility of the property precludes the possibility of its commercial importance.

**MOLYBDENITE NEAR CORONA, RIVERSIDE COUNTY.**

An occurrence of molybdenite about  $4\frac{1}{2}$  miles northeast of Corona in a granite quarry operated by John Fletcher has been noted by Hess.<sup>a</sup> Specimens submitted to the bureau show the mineral to occur in a medium-grained gray granite in flakes up to an inch in diameter. A piece of typical ore is illustrated in Plate XI, *B.* Hess, who visited the deposit in 1907, states that: "The granite is cut by thin pegmatite dikes one-half inch to 2 inches wide. Molybdenite in flakes up to one-half inch across accompanies the dikes in small quantity. In one or two places small flakes may be found in the granite for a distance of 2 or 3 inches from the dikes. The only other metallic mineral found in the dike is iron pyrites in which an assay is said to have shown some copper, gold, and silver." The deposit is said to be small and is probably not of economic importance.

**MOLYBDENITE NEAR CALIFORNIA HOT SPRINGS, TULARE COUNTY.**

Molybdenite in small flakes up to about one-fourth inch maximum diameter occurs in milky quartz in a large vein about one-half mile from California Hot Springs. This town is about 22 miles west of Ducor, the nearest point on the Southern Pacific Railroad, from

<sup>a</sup> Hess, F. L., Some molybdenum deposits of Maine, Utah, and California; Contributions to Economic Geology: U. S. Geol. Survey Bull. 340, 1907, p. 238.

which it is reached by a good automobile road. The deposit is situated in sec. 31, T. 23 S., R. 31 E., at an elevation of about 3,300 feet, and is owned by L. S. Wingrove, of California Hot Springs. Practically no development work has been done on the vein, but a hasty examination of a considerable part of the outcrop indicated to the writer that the occurrence of molybdenite, even as minute specks, is so occasional that there seems to be no possibility of the deposit being of commercial importance.

#### MOLYBDENITE AT CALIENTE, KERN COUNTY.

J. B. Ferris of Caliente has submitted to the bureau specimens of low-grade gold ore carrying molybdenite in finely disseminated grains, coming from the Golden group of claims a few miles south of the town. The ore body is large, but as determined by samples taken by the author from several places in the tunnels and the drifts by which it is developed, it is so poor in molybdenite as to probably be of no commercial importance as a source of that mineral.

#### COLORADO.

Molybdenite is widely distributed throughout Colorado, its occurrence having been noted in almost every mountain county. The principal deposits visited by the author are at Red Mountain near Empire in Clear Creek County, and near Climax and Breckenridge in Summit County. A description of these deposits, together with notes as to some of the other occurrences listed in the accompanying table, are given in the following pages.

Several occurrences of wulfenite in the State have been reported, but so far as known these are only of mineralogical interest.

#### *Occurrences of molybdenite in Colorado.*

Location of deposit.	Remarks.	Reference.
Chaffee County:		
Salida.....	In quartz.....	Specimen in Colorado State Museum.
South of Fisher.....	See p. 70.....	
Near Buena Vista.....	See Pl. II, A.....	Do.
Clear Creek County:		
Red Mountain near Empire.....	See p. 64.....	
Conejos County:		
At Platoro.....	In spherical forms with concentric structure.	Dana, E. S., A system of mineralogy, 6th ed., 1911, p. 1042.
Custer County:		
Grape Creek.....	In decomposed basic rock with calcite, chlorite, and pyrite.	Specimen in U. S. National Museum.
12 miles north of Westcliffe.....	See p. 71.....	
Eagle County:		
Near Redcliff.....	.....	
Fremont County:		
Near Rito Alto Peak.....	See p. 70.....	
7 miles south of Parkdale.....	See p. 70.....	
Gilpin County:		
From Apex.....	With molybdite.....	Specimen in Colorado State Museum.

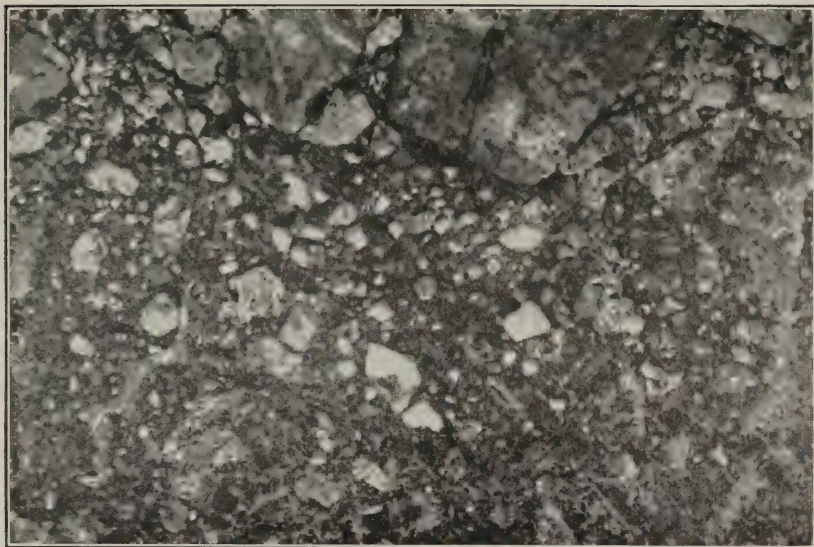
*Occurrences of molybdenite in Colorado—Continued.*

Location of deposit.	Remarks.	Reference.
Gunnison County:		
4 miles north of Pitkin near Rock Creek.	.....	Mineral Industry, vol. 6, 189 p. 485.
At the head of Crystal River near Marble.	.....	Letter from J. C. Hersey, Leadville, Colo.
Near Tin Cup.....	Crystalline molybdenite in white quartz.	Specimens in collection of U. Geol. Survey.
2 miles from Pitkin.....	In quartz veins.....	U. S. Geol. Survey Bull. 58 1914, p. 46.
8 miles southeast of Marble.....	See p. 71.....	
Huerfano County:		
2 miles southeast of Mosca Pass.	Large flakes in float.....	Letter from John O'Toole, Denver, Colo.
Lake County:		
In Maid of Erin mine at Leadville; also near the head of Big Evans Gulch.	.....	Eng. and Min. Jour., vol. 8 1905, p. 127.
Larimer County:		
4 miles south of St. Cloud.....	See p. 72.....	
Ouray County:		
Engineer Mountain.....	.....	Letter from E. C. Weatherly, Ouray, Colo.
Pitkin County:		
South of Red Mountain.....	See p. 71.....	
In Lincoln Gulch southeast of Roaring Fork of the Grand River.	In silicious gangue.....	Letter from J. C. Hersey, Leadville, Colo.
San Juan County:		
Between Maggie and Picayune Gulches and Red Peak.	With copper, nickel, and bismuth minerals.	Trans. Am. Inst. Min. Eng., vol. 21, pp. 189-190.
Near Silverton.....	.....	U. S. Geol. Survey Bull. 58 1914, p. 46.
San Miguel County:		
1 mile east of Ophir.....	See p. 69.....	
Summit County:		
On Bartlett Mountain.....	See p. 68.....	
On Chalk Mountain.....	See p. 68.....	
Fremont Pass.....	.....	U. S. Geol. Survey Bull. 58 1914, p. 46.
Near Kokomo.....	See p. 72.....	
2 miles north of Kokomo.....	See p. 72.....	
Lenawee mine, between Kingston and Montezuma.	.....	Specimen in courthouse, Breckenridge.
Southern slope of Quandary Mountain.	See p. 67.....	
Teller County:		
Cripple Creek.....	.....	

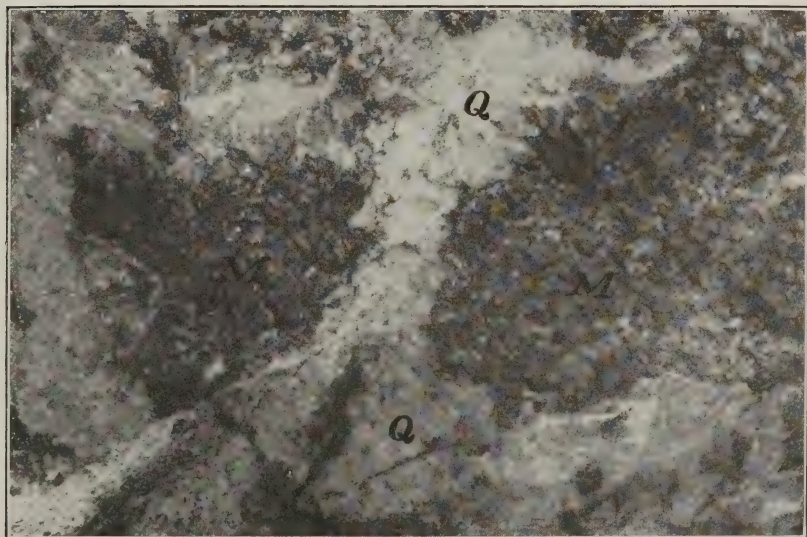
**MOLYBDENITE MINE OF PRIMOS CHEMICAL CO., NEAR EMPIRE**

The molybdenite mine of the Primos Chemical Co., near Empire, Clear Creek County, consists of a group of nine claims situated on the eastern slope of Red Mountain, at an altitude of about 10,500 to 12,000 feet. The property is reached from Empire, a station on the Georgetown branch of the Colorado & Southern Railroad, by a good wagon road 14 miles long. The ore-bearing bodies consist of three veins of low-grade ore, the general strike of which, as far as determined by present development work, lies between east by west and northeast by southwest, across the ridge of the mountain. The strikes of the individual veins show a convergence in a westerly direction. Their dip, as indicated by the position of the outcrops with reference to their intersection by a tunnel several hundred feet below, is 50° to 60° in a general northwestern direction. The veins are named from the southeast, and are known as Veins No. 1, No. 2, and No. 3. The ore zone in general; that is, the ground included





A. SECTION THROUGH TYPICAL MOLYBDENITE ORE FROM PRIMOS MINE, NEAR EMPIRE, COLO. (NATURAL SIZE.)

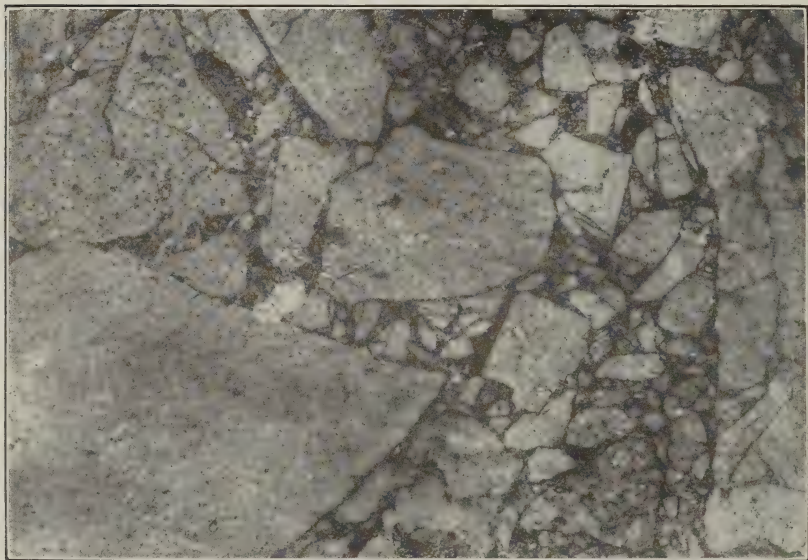


B. TYPICAL MOLYBDENITE ORE FROM BARTLETT MOUNTAIN, SUMMIT COUNTY, COLO. (NATURAL SIZE.) M, MOLYBDENITE; Q, QUARTZ.



A. VIEW OF RED MOUNTAIN, NEAR EMPIRE, COLO.

The arrow indicates the waste dump at the mouth of the upper tunnel of Primos Mine.



B. SECTION THROUGH TYPICAL MOLYBDENITE ORE FROM GREAT WESTERN MINE, NEAR CHICO, PARK COUNTY, MONT.

between the footwall of Vein No. 1 and the hanging wall of Vein No. 3, where cut by the tunnel, is about 200 feet wide. The veins vary greatly in width, both among themselves and along their strikes. Further, they are not particularly well defined, the ore running off into the walls in the form of thin veinlets and stringers.

Vein No. 1 has not been sufficiently developed to warrant even an approximate statement of its average width, but at its intersection by the upper tunnel it is 3 to 4 feet across. Vein No. 2, as exposed by about 1,000 feet of development work, shows a width varying from 3 or 4 inches to 4 or 5 feet, and it has an average width of perhaps 2 feet. Vein No. 2, at its intersection by the upper tunnel, showed 10 feet of ore.

The ore occurs in two forms—one consists of an alaskite (?) breccia in which the molybdenite occurs as occasional small flakes but more often in finely granular form associated with iron pyrite in the interstices of the breccia; in the other the molybdenite occurs in small veinlets and stringers running off into the alaskite-porphry (?) country rock. The brecciated type of ore constitutes the vein material proper, and is well illustrated by Plate XII, A. Oxidation of the molybdenite along the outcrops of the veins has resulted in the formation of considerable molybdite, but the latter mineral disappears entirely with depth. It is stated that the ore contains no copper or other deleterious elements.

The author first visited the property in July, 1913, when development work consisted of only two or three shallow open cuts along the outcrop of one vein. The existence of ore bodies was evidenced, however, by large masses of vein material exposed along the outcrops and by the great quantities of float found throughout the rock slides on the mountain slope below. One of these large masses of outcropping vein material, which showed considerable molybdenite here and there over its entire surface, and was colored in many places by yellow molybdite, is pictured in Plate X, B.

Development of the property was started in earnest in November, 1914, when the present owners obtained control, and to August, 1915, work had been pursued at the rate of about 350 feet per month. At the latter date development consisted of an upper working with 2,500 to 3,000 feet of tunnel, drifts, and crosscuts at two-thirds the distance to the top of the mountain and at an altitude of about 11,800 feet; and also of a tunnel which had just been commenced about 600 feet lower down. The situation of the upper workings is well shown in Plate XIII, A.

The upper workings consist roughly of a main tunnel running in a northwesterly direction for about 500 feet, and of a drift along Vein No. 2 about 800 feet west. Twenty-one crosscuts up to 20 feet in



length have been made along this drift, usually at intervals of about 30 feet, but occasionally at 15-foot intervals. A stope has been started on this vein, and in July, 1915, it had been opened for a horizontal distance of about 40 feet and mined to a height of about 20 feet, the ore having an average width of about  $4\frac{1}{2}$  feet. A second and larger stope has been begun to the north of Vein No. 2. The ore body at this point is thought to be either separated from the vein by a large "horse," or to lie on the intersection of veins No. 2 and No. 3. This stope has been mined for a horizontal distance of about 130 feet and to a height of 40 to 45 feet; in width it varies from 6 to 15 feet.

Only a few feet of work has been done on vein No. 1 as the showing of ore encountered at its intersection by the tunnel was poor. Vein No. 3 has been drifted on for about 20 feet on either side of the tunnel, and 10 feet of good ore is said to have been encountered in these drifts.

As already stated, work on the lower tunnel was commenced in August, 1915. Assuming that the dip of the veins is  $55^\circ$  and that they will be encountered on this level, over 1,000 feet of backs will be developed by this lower tunnel.

Stoping is carried on largely by hand, as the ground is fairly soft and easily worked. Air drills, however, are used in running the tunnels and drifts.

The ore is trammed to a sorting house at the mouth of the upper tunnel and dumped into a bin from which it is drawn by the sorters as needed. Ore from the bin is screened on small 2-mesh hand sieves. The undersize through the screens is saved and the oversize hand sorted. The rejected oversize is thrown into a car which carries it to the waste dump, and the selected ore is sacked. Filled sacks weigh 75 to 125 pounds each, according to the size and richness of the ore. These are packed down the mountain on mules and stacked near the wagon road. At the time of the author's visit, in July, 1915, the pack train consisted of 14 animals, each of which carried three to four sacks of ore. At the base of the mountain the sacks are loaded onto wagons, weighed, and hauled to the railroad station at Empire, where their contents are emptied into freight cars for shipment to the Eastern plant of the company. From January 1, 1915, to August, 1915, shipments are reported to have averaged two to three carloads of  $22\frac{1}{2}$  tons each per week.

The mine has not been in operation for sufficient time and the ore bodies are not well enough developed to justify any statements as to mining cost.

A combined bunk and compressor house, which also contains an office, dining-room and quarters for cooking, is situated about 1,300 feet below the mouth of the upper tunnel and close to the wagon





### MAP OF PART OF SUMMIT COUNTY, COLORADO,

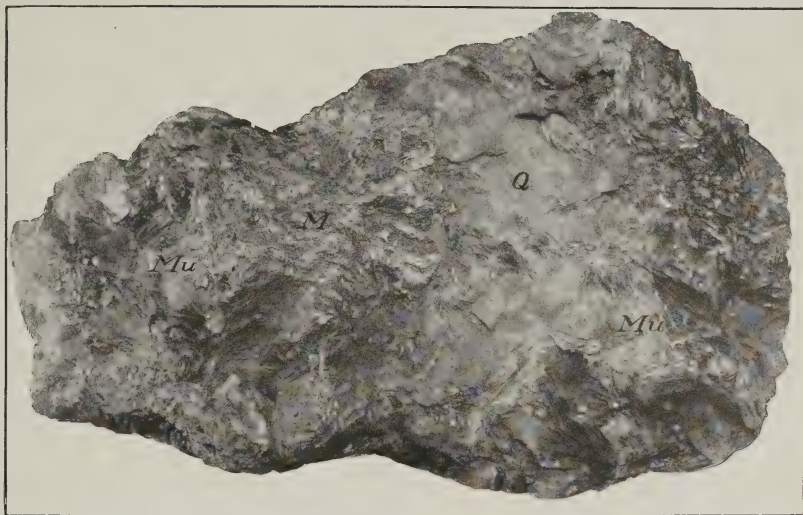
SHOWING LOCATION OF MOLYBDENITE DEPOSITS ON QUANDARY, BARTLETT, AND CHALK MOUNTAINS

Scale  $\frac{1}{125000}$

1 2 3 4 5 Miles

Contour interval 100 feet.

LIBRARY  
OF THE  
UNIVERSITY OF ILLINOIS



A. TYPICAL MOLYBDENITE ORE FROM SALAMANDER CLAIM, QUANDARY PEAK, SUMMIT COUNTY, COLO. (TWO-THIRDS NATURAL SIZE.) *M*, MOLYBDENITE; *Q*, QUARTZ; *Mu*, MUSCOVITE.



B. SCRAPING TAILINGS FOR RETREATMENT TO RECOVER WULFENITE, MAMMOTH, ARIZ.

LIBRARY  
OF THE  
UNIVERSITY OF ILLINOIS



road. The compressor room in this building contains a 60-horsepower boiler which is fired with wood; it supplies steam for the operation of an Ingersoll-Rand "Imperial" air compressor, type 10. The two steam cylinders on the compressor are 13 by 10 inches and 8 by 10 inches, and the corresponding air cylinders are 14 by 10 inches and 7½ by 10 inches. Although it is rated as a three-drill machine it will operate only two drills at this altitude.

Wood for the boiler is obtained by cutting standing timber in a burned-over area of forest in the vicinity. It is hauled to a small sawmill situated a few hundred yards south of the boiler house, and cut into 4-foot lengths. Stables have been built close to the bunk and compressor house. At the upper tunnel level there is a bunk house and blacksmith shop and the ore-sorting house already referred to.

The production of this property to date has been probably 1,500 to 2,000 tons of ore, averaging approximately 2 per cent molybdenum. In opening the property during the winter months many difficulties were encountered, owing to the altitude and to winter storms.

#### **SALAMANDER AND BLUE VALLEY MOLYBDENITE CLAIMS, NEAR BRECKENRIDGE, SUMMIT COUNTY.**

There is an interesting occurrence of molybdenite about 11 miles southwest of Breckenridge, Summit County. The deposit is on the south slope of Quandary Mountain at an altitude of about 12,000 feet (see Pl. XIV), and consists of two pegmatite veins which contain the molybdenite, and on which are two claims known as the Salamander and the Blue Valley. These claims are reached from Breckenridge by a good wagon road for 9 miles up the valleys of Blue River and Monte Christo Creek, thence continuing up the valley of the latter stream for 1½ miles by a poor mountain road, and then across the creek and up the mountain side for half a mile by trail.

At the time of the author's visit to the deposit in October, 1913, a violent snowstorm was in progress, and for that reason his examination was necessarily incomplete.

The pegmatite veins already referred to consist largely of muscovite and quartz with some feldspar, and carry molybdenite and chalcopyrite as accessory minerals, as well as small quantities of molybdenite developed by alteration of the molybdenite in places where the latter has been exposed to the air for a considerable time. The veins, which appear to be nearly vertical, traverse a dark-gray micaceous schist or gneiss composed essentially of quartz, muscovite, biotite, and feldspar, and strike in a general northerly direction.

A specimen of typical ore from the Salamander claim is illustrated in Plate XV, A. A sample of about 100 pounds of picked ore, which the author selected as representative of the general grade of concentrating ore that this claim could produce, on analysis showed 4.35

per cent molybdenite and 0.45 per cent copper. The molybdenite occurs in coarsely crystalline aggregates up to  $1\frac{1}{2}$  inches in maximum diameter, but on account of its association with chalcopyrite and with such a large percentage of muscovite, it probably could not be recovered in a high-grade concentrate without a rather complex treatment.

On the Salamander claim a tunnel has been run for a distance of 20 feet on the vein, which at this point has a width of 3 to 4 feet.

On the Blue Valley claim there is only a small showing of molybdenite, and no development other than the putting in of several shots at various places along the outcrop. The vein averages only 4 to 6 inches in width and gives no promise of any commercial production of ore.

Directly down the mountain from the claims there is a small lake, and one-half mile farther up the gulch a larger lake at an elevation of 300 or 400 feet above the one first mentioned. Ample water power for all mining purposes could easily be developed from this upper lake, and there is sufficient timber in the immediate neighborhood for all prospective mining needs.

#### **MOLYBDENITE ON BARTLETT MOUNTAIN, SUMMIT COUNTY.**

There is a large low-grade deposit of molybdenite ore above timber line at an elevation of approximately 12,000 feet on the southwestern slope of Bartlett Mountain, Summit County. The deposit is about two miles southeast of Robinson, and about a mile and a quarter east of Climax, stations on the Colorado & Southern Railway. Its location is shown in Plate XIV. At the time of the writer's visit to the property in the late fall of 1913 it was covered with snow and little first-hand information concerning the deposit could be obtained. In places where the snow had blown off large quantities of slide rock were exposed, which on an average appeared to contain 0.5 to 1 per cent molybdenite, the mineral occurring in the cracks and fractures of a crystalline quartz, most of which was associated with more or less pyrite, but was practically free from copper minerals. Although most of the deposit was covered with snow, the author had no difficulty in picking up from this particular slide within a few minutes 100 pounds of rock which on sampling and analysis yielded 1.06 per cent  $\text{MoS}_2$ . A specimen of typical ore is shown in Plate XII, *B*.

Eight adjoining claims are reported to have been located on the deposit. The chief development work at the time of the writer's visit consisted of two tunnels, one near the northwestern end of the deposit, on the claims of H. Leal, and the other near the center of the deposit, on the Gillaspey claim, known as "Denver No. 2." The first tunnel had been driven in an eastern direction for a distance of

over 500 feet, and was being pushed ahead with the idea of cross-cutting the deposit. As far as work had progressed the character of the rock seemed essentially the same, and the writer was told that samples taken at frequent intervals throughout the entire length of the tunnel showed an average content of 0.7 per cent  $\text{MoS}_2$ . The second tunnel was said to be 30 to 40 feet long, but as its portal was closed by a snowdrift it was not entered.

The following notes concerning recent development on the deposit were kindly furnished the writer by Ira Snyder, of Denver, Colo., who has purchased the "Denver No. 2" claim:

The tunnel on the Leal claims has been extended to a total length of 800 feet and at the face a 100-foot drift runs at right angles to the line of the tunnel. No walls have been encountered and there has been no perceptible change in the character or average richness of the ore. A lease is said to have been taken on the Leal claims by Messrs. McDonald, Noble, and King, of Leadville, who have built a wagon road to the property from Climax. These men are also reported to have mined several hundred tons of ore from the claims of Charles J. Senter, near the southeastern end of the deposit, shipping the same to Leadville for concentration by flotation in an experimental plant. The tunnel on the "Denver No. 2" is stated to be now about 75 feet in length, and the ore encountered is reported to contain an average of about 1.3 per cent  $\text{MoS}_2$ . Experiments on a large scale are being carried on in Denver for concentrating by flotation the ore from this claim. The results of these tests have been satisfactory, and preparations are being made for the installation of a plant at the mine during the coming summer.

#### MOLYBDENITE NEAR OPHIR, SAN MIGUEL COUNTY.

The Molybdenite Queen claim, owned by G. C. Knox, of Denver, and J. Belisle, of Norwood, Colo., is situated in the Iron Spring mining district about 1 mile southeast of the town of Ophir, which is  $2\frac{1}{2}$  miles east of Ophir Loop, a station on the main line of the Rio Grande Southern Railroad. It crosses Nevada Gulch about 600 feet above the creek at the base of the hill and adjoins the Favorite group of gold claims. The deposit is said to consist of a well-defined vein of white quartz 20 to 25 feet wide, traversing a poryphyry country rock. The molybdenite occurs in the vein in flaky crystal aggregates and irregular-shaped masses from the size of a pea to pieces three-quarters of an inch or more in diameter. In a 25-pound lot of ore, said to be a sample taken on the surface across the width of the vein, the writer would estimate the molybdenum sulphide content to be 2 to 3 per cent. In this sample considerable alteration of the molybdenite to molybdite had taken place and a small quantity of pyrite was noticed. The ore appears of such character as to be readily concentrated by either electrostatic or flotation methods.

The vein is said to strike in a general southwestern direction, but its dip has as yet not been determined. Practically no development work has been done on the outcrop, but the vein is reported to be cut at a depth of about 600 feet by the Deadwood tunnel and to be approxi-



mately 20 feet wide at the place of its intersection. However, the owners of the property have been unable to verify this statement, as entrance to the tunnel can not be had, owing to a cave-in only a few feet from the portal.<sup>a</sup>

#### OTHER OCCURRENCES OF MOLYBDENITE IN COLORADO.

##### DEPOSIT NEAR ROBINSON, SUMMIT COUNTY.

A considerable deposit of low-grade molybdenite ore, practically identical with that of Bartlett Mountain described above, is situated on Chalk Mountain near Robinson, Summit County. (See Pl. XIV.) Four claims located on the deposit are owned by E. G. Heckendorf, J. W. Harris, and S. H. Weber, of Denver, and J. A. Weber, of Leadville, Colo. The ore body is reported to be 30 to 40 feet wide, and its outcrop, which runs northeast, is said to have been traced for over 3,000 feet. In October, 1915, development work consisted of seven open cuts, samples from which are reported to have assayed 1 to 3 per cent of molybdenum disulphide.

##### DEPOSIT NEAR NATHROP, CHAFFEE COUNTY.

Bert Bergstrom and H. C. Hayes, of Buena Vista, Colo., are said to have opened a promising molybdenite property in Chaffee County, a few miles south of Fisher, a station on a branch of the Colorado & Southern Railroad from Nathrop. It is understood that the molybdenite occurs in quartz in flakes up to an inch in maximum diameter and that the ore is practically free from copper minerals. About 1 ton of high-grade molybdenite is reported to have been obtained by careful cobbing and hand sorting of some of the richer ore. It is said that development work on the property is going ahead rapidly.

##### PROSPECT NEAR PARKDALE, FREMONT COUNTY.

Ralph Fairchild, of Canon City, Colo., has sent the Bureau of Mines specimens of molybdenite ore that is said to occur in considerable quantity in a prospect being developed for copper 7 miles south of Parkdale, Fremont County. The specimens show 5 to 6 per cent of molybdenite associated with pyrite and chalcopyrite in a siliceous gangue. Development work on the property consists of a shaft about 100 feet deep.

An occurrence of molybdenite is reported in Fremont County on the northern slope of Rito Alto Peak in the Sangre de Christo Range. The deposit is above timber line, at an altitude of over 11,000 feet, and is difficult of access, being 18 or 20 miles from Cotapaxi, the nearest station on the Denver & Rio Grande Railroad, and the last 6 or 8 miles of the journey to the property is over a very rough

<sup>a</sup> The above information is from a report given the writer by G. C. Knox, of Denver, Colo., on Nov. 8, 1915.



trail. It is understood that the only development work consists of a shaft about 30 feet deep. Specimens of ore said to have come from this occurrence and sent to the writer by B. J. Hillman, of Denver, Colo., consist of a quartzose rock in which the molybdenite occurs as flakes and irregular-shaped masses up to one-half inch or more in diameter and constitutes perhaps 4 or 5 per cent of the whole. The molybdenite is associated with chalcopyrite. No information regarding the size of the deposit nor the general grade of the ore is available.

#### SPECIMENS FROM WESTCLIFFE, CUSTER COUNTY.

Specimens of molybdenite ore said to be representative of a considerable quantity of material on the dump of an old mine 12 miles north of Westcliffe, Custer County, have been submitted to the bureau by L. H. Schoolfield, of that town. The rock in these specimens is so altered that its identification is impossible. It appears to be a highly metamorphosed calcareous sediment, and, besides the molybdenite, which occurs in flakes up to one-fourth inch in maximum diameter, it contains siderite, calcite, quartz, and some feldspar. It is seemingly free from copper. The writer estimates the molybdenite content of the samples submitted as 2 or 3 per cent.

#### CLAIM NEAR MARBLE, GUNNISON COUNTY.

The Bureau of Mines has received two samples of molybdenite ore said to have come from a claim recently located by John O'Toole, of Denver, on the southwestern slope of Treasury Mountain, about 8 miles north of Marble, Gunnison County. One sample consists of milky quartz, in which the molybdenite, associated with considerable molybdenite, occurs in flakes up to one-quarter of an inch in diameter. In the other sample the quartz is crystalline and the molybdenite more granular in character. Both lots of ore were free from copper minerals and appeared to contain 2 to 3 per cent of molybdenum disulphide. The ore was said to have been taken from two roughly parallel veins about 150 feet apart and striking northeast. The average width of the larger vein was stated to be about 3 feet and that of the smaller about 20 inches.

#### DEPOSIT SOUTH OF ASPEN, PITKIN COUNTY.

Molybdenite is reported to occur in a deposit owned by F. E. Kendrick and associates, of Leadville, Colo., and situated south of Red Mountain about 20 miles south of Aspen, Pitkin County. The vein, which is called the Green Horn lode, is said to be 6 feet wide. A specimen of ore purported to have come from this deposit consists of a brecciated quartz heavily impregnated with pyrite and containing occasional fine flakes of molybdenite.

## CLAIMS NEAR KOKOMO, SUMMIT COUNTY.

G. M. Bacom, of St. Elmo, Colo., has submitted to the bureau samples of molybdenite ore said to have come from a claim near Kokomo, Summit County. The ore consists of white crystalline quartz containing a small amount of fine-grained molybdenite and its alteration product, molybdite. It seems identical with the molybdenite ores from Bartlett and Chalk Mountains which are described above. No information regarding the exact location or size of the deposit is available.

Small quantities of flaky molybdenite occur in ore from the tunnel on claims belonging to Joseph Bryant, of Kokomo, Summit County, and situated about 2 miles southeast of that town. The ore consists principally of quartz heavily impregnated with pyrite.

## PROPERTY NEAR ST. CLOUD, LARIMER COUNTY.

E. F. Bartlett, of St. Cloud, Larimer County, has submitted specimens of magnetite containing occasional patches of molybdenite. This ore is said to have come from a property about 4 miles south of the town.

## MONTANA.

Out of a dozen or more authenticated occurrences of molybdenum ore in Montana, only two occurrences of molybdenite and one of wulfenite have been examined by the writer. Unfortunately, these deposits were of necessity visited during the late fall when they were covered with snow, which precluded an examination of their outcrops and otherwise hampered their investigation. Besides the occurrences of molybdenite listed in the accompanying table, two occurrences of wulfenite have been reported in the State. One of them is at Radersburg,<sup>a</sup> in Broadwater County, and the other is near Twin Bridges, in Madison County. The latter deposit was visited and is described hereafter.

*Occurrences of molybdenite in Montana.*

Location.	Remarks.	Reference.
Beaverhead County: Head of Birch Creek on Tent Mountain, 12 miles west of Apex and 30 miles northwest of Dillon.	In quartz.....	Mineral Ind., vol. 13, 1904, p. 328, Mineral Resources U. S., 1903, U. S. Geol. Survey, 1904, p. 308, and Mineral Resources, U. S., 1904, U. S. Geol. Survey, 1905, p. 341.
Wise River.....	See p. 76.....	Letter from F. W. Hall, Wise River, Mont.
Broadwater County: At Winton.....	With calcite and vesuvianite.....	Specimen in U. S. National Mu- seum.
Fergus County: At Neihart.....	.....	U. S. Geol. Survey Bull. 585, 1914, p. 112.

<sup>a</sup> Specimen in Harvard collection, Agassiz Museum.

*Occurrences of molybdenite in Montana—Continued.*

Location.	Remarks.	Reference.
Jefferson County:		
At Clancey.....	With feldspar, quartz, chalcopyrite, and muscovite.	Specimen in U. S. National Museum.
Near Homestake.....		Mineral Resources U. S., 1906, U. S. Geol. Survey, 1907, p. 528; Eng. and Min. Jour., vol. 76, 1903, pp. 370, 791.
Madison County:		
Near Sheridan.....		Mineral Resources U. S., 1906, U. S. Geol. Survey, 1907, p. 528.
Missoula County:		
Leslie Copper Mine in western part of county.	In quantity on 150-foot level....	Mineral Resources U. S., 1901, U. S. Geol. Survey, 1902, p. 266.
Park County:		
Near Chico Hot Springs.....	See p. 73.....	
Near Cooke, at head of Stillwater River.	See p. 76.....	
Powell County:		
In Carpenter Gulch, 2 miles east of Ophir.	See p. 75.....	Mineral Resources U. S., 1903, U. S. Geol. Survey., 1904, p. 308, and Mineral Resources U. S., 1904, U. S. Geol. Survey. 1905, p. 341.
Silverbow County:		
In the Montgomery, Altoona, and neighboring claims and number of places east of the Flat and in the Gagnon mine.		Weed, W. H., Geology and ore deposits of the Butte district, Montana: U. S. Geol. Survey Prof. Paper 74, 1912, p. 79.

**MOLYBDENITE NEAR CHICO HOT SPRINGS, PARK COUNTY.**

The molybdenite property of the Great Western Mining & Milling Co. is situated in Park County about 6½ miles from Chico Hot Springs and 5½ miles southeast of Chico. It is at an elevation of about 7,500 feet on the northeast bank of Emigrant Creek opposite Emigrant Peak, and on the southwest slope of the ridge joining Chico Peak and Mineral Mountain, outlying peaks of the Absaroka Range.

The property is about 10 miles from Emigrant, the nearest railroad point on the Yellowstone Park branch of the Northern Pacific Railroad from which it is reached by wagon road via Chico Hot Springs and Chico. From Emigrant to Chico this road is in fair condition and there are no bad grades, but up Emigrant Creek the road rises almost continuously and many of the grades are so steep that the hauling of even moderate loads is accomplished with difficulty.

The deposit consists of a vein of what is seemingly a brecciated and somewhat altered fine-grained light-gray porphyry in which fine-grained molybdenite intimately associated with pyrite forms the matrix and permeates the individual fragments of brecciated material. Plate XIII, *B*, illustrates a cross section of typical ore from the deposit. According to Iddings and Weed<sup>a</sup> the rocks downstream from the mine consists of basic andesitic breccia and flows, whereas above the mine they are acid porphyrites, andesite, and dacite. At the time the writer visited the property, in the late fall of 1913, the outcrop of the vein was covered with snow and the underground

<sup>a</sup> Iddings, J. P., and Weed, W. H., Livingston folio (No. 1), Geol. Atlas U. S., U. S. Geol. Survey, 1894.



workings were inaccessible owing to the presence of a great quantity of carbonic acid gas. As the writer was unable to make an examination of the workings, H. F. Lawrence, the mine manager, kindly supplied the following notes:

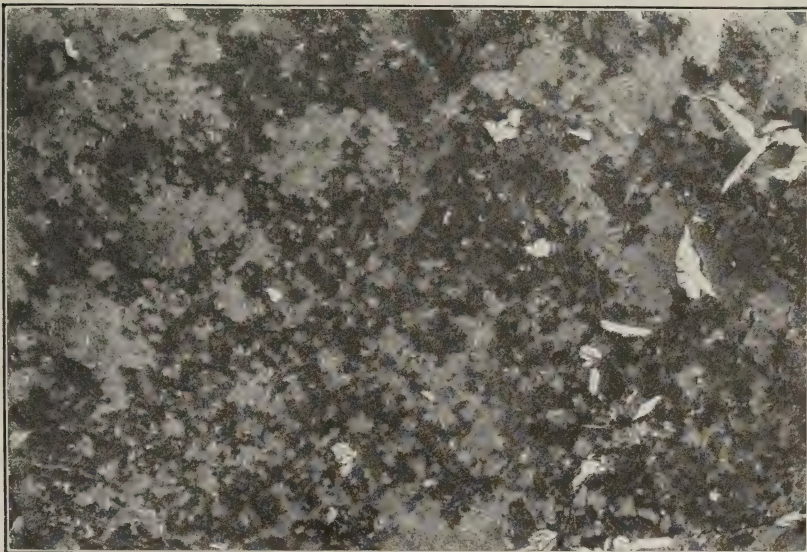
The molybdenite deposit was cut many years ago by a tunnel driven in search of gold. This old tunnel (said to be 700 to 800 feet long but the face of which is inaccessible owing to cave-ins) intersects the molybdenite-bearing vein 360 feet from the portal and at a vertical depth of about 260 feet below the outcrop of the vein. At the point of intersection by the tunnel the vein is about 14 feet wide and so nearly perpendicular that it is impossible to say which is the foot and which the hanging wall. Only one wall is well defined. There is practically no development work on the vein at the tunnel level, only a few feet of drifting having been done and a raise started. The strike of the vein is approximately N. 30° W. by S. 30° E. It has been traced on the surface for over 1,000 feet by means of open cuts, and float ore indicates that it continues much farther. The average width of the outcrop is 12 to 15 feet.

As regards the molybdenite content of the ore, the author was informed by E. C. Sackett of Livingston, Mont., that two samples taken by him across the drift in the tunnel assayed 14 and 23 per cent  $\text{MoS}_2$ , respectively, but these samples were probably taken in the richest places that could be found. As indicated by several tons of ore in a small mill erected for experimental concentration work near the mouth of the tunnel, the ore in the drift is of good grade, averaging perhaps 5 to 6 per cent  $\text{MoS}_2$ . As far as shown by tests in a few small specimens, the ore contains no copper.

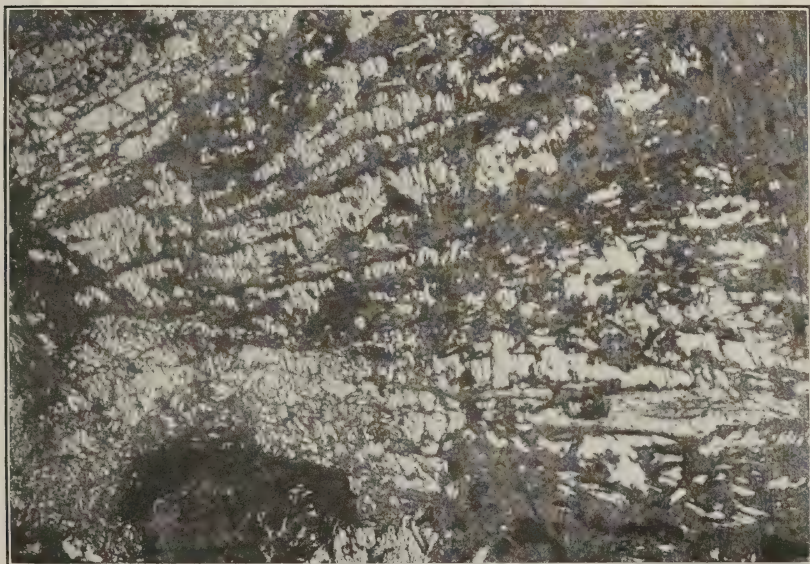
At the time of the writer's visit 10 to 12 men were employed in gathering a fuel supply for winter. A new boarding house had just been completed and the foundation for a mill erected, and active development work was contemplated within a few weeks. It was proposed to treat the ore by water flotation without the use of acid or oil on machines designed by George B. Allison, president of the company. Whether the proposed process of treatment will be successful can not be foretold, but from a number of concentration tests on a model machine, which the writer saw made at Butte, Mont., it appears to be impossible to obtain a high-grade concentrate (85 per cent or more  $\text{MoS}_2$ ) from the ore by the process contemplated. The association of the molybdenite with the gangue is intimate and it is stated that the ore must be crushed through a 60-mesh to 80-mesh screen before the molybdenite is liberated. It is practically impossible to feed this fine material to the water surface in an ore sheet only one mineral particle in depth, and for this reason considerable fine silica is supported on the floating film of molybdenite. A part of the pyrite in the ore also floats, and both of these impurities detract considerably from the grade of the concentrates.

Plenty of wood for both fuel and mine timbers can be obtained in the immediate vicinity of the mine, and Emigrant Creek carries sufficient water to supply a large mill.





A. SECTION THROUGH MOLYBDENITE ORE FROM QUIGLEY MINE, CARPENTER GULCH, NEAR OPHIR, POWELL COUNTY, MONT. (SLIGHTLY ENLARGED.)



B. SECTION THROUGH MOLYBDENITE ORE FROM ROMERO MINE, PORVENIR, SAN MIGUEL COUNTY, N. MEX. (TWO TIMES NATURAL SIZE.)

LIBRARY  
OF THE  
UNIVERSITY OF ILLINOIS

Inquiries addressed to the president of the company as to the progress made in development work have not been answered; hence the author can not state what the result of this work was or whether it was ever carried out as contemplated.

#### MOLYBDENITE IN CARPENTER GULCH, POWELL COUNTY.

There is an occurrence of molybdenite ore about 2 miles east of Ophir, in Carpenter Gulch, Powell County. The property, which is owned by Thomas L. Quigley, of Sacramento, Cal., is about  $7\frac{1}{2}$  miles from Avon and 12 miles from Elliston, stations on the Northern Pacific Railroad, from which it is reached by wagon road. The elevation of the mine is approximately 5,700 feet. The author visited the property in the late fall of 1913, when the outcrop of the deposit was covered with snow. Apparently the property had not been worked for several years, and the portal of the tunnel by which it was developed was locked. Furthermore, the tunnel had caved badly near the portal, so that access to it could not be had. By clearing away the snow from a part of the dump the writer was able to obtain 30 or 40 pounds of ore, containing perhaps 2 or 3 per cent of molybdenite in flakes and irregular-shaped masses up to an inch in maximum diameter. The rock in which the molybdenite occurs is seemingly a metamorphosed limestone that has undergone various stages of alteration. Some specimens are much more highly altered than others. The great differences in the character of individual specimens of the ore are illustrated by Plate XVI, A, which also plainly shows how the molybdenite occurs. The most prominent minerals in the rock are garnet, epidote, quartz, some unidentified silicates, and one or more calcareous carbonates. The ore is seemingly free from copper minerals.

Tests of the crushed ore show that the garnet contained in it is a good conductor of electricity, and therefore concentration by electrostatic methods is impracticable, but the ore is evidently suitable for treatment by flotation. From the size of the dump at the tunnel mouth the writer would estimate that the tunnel had been projected between 150 and 200 feet, but as the tunnel was inaccessible for the reasons stated above nothing definite can be said here as to the size of the deposit and of the possibilities of its being of commercial importance.

A second occurrence of molybdenite in Carpenter Gulch is reported at the head waters of the creek, about  $2\frac{1}{2}$  miles above the occurrence already noted.<sup>a</sup> Specimens said to have come from this deposit show the molybdenite occurring in small flakes in what is seemingly a metamorphosed limestone.

---

<sup>a</sup> Letter, dated Mar. 6, 1915, from C. L. Smith, Deer Lodge, Mont.



**OTHER MOLYBDENITE OCCURRENCES IN MONTANA.**

The Bureau of Mines has received from G. H. Gibson of Cooke, Mont., several samples of molybdenite ore said to have come from a deposit on the headwaters of the Stillwater River near Cooke, Park County. The samples consist of greenish-black, medium to coarse grained garnetiferous amphibolite in which the molybdenite occurs in small irregularly distributed patches. The rock is heavy and tough, and at least 75 per cent of it is made up of dark-green hornblende. The other principal constituents are a reddish-brown garnet and quartz, the latter occurring for the most part in narrow veinlets. Pyrite occurs associated with the molybdenite both in the hornblende and in the garnet. In the 30 or 40 pounds of material submitted to the bureau the author would estimate the molybdenite content to be 1 to 1.5 per cent.

B. R. Holland of Cooke, Mont., has sent in specimens of quartz containing small flakes of molybdenite associated with molybdate. These are said to have come from a prospect in the vicinity of that town.

Specimens of molybdenite ore, said to have been obtained from stringers and small veins cut in driving a tunnel in Comet Mountain on the property of the Boston-Montana Development Co., Beaverhead County, have been received from F. W. Hall, of Wise River, Mont. The molybdenite occurs as small flakes in quartz and is stated to occur in quantity.<sup>a</sup>

**WULFENITE ORE NEAR TWIN BRIDGES, MADISON COUNTY.**

The property of the Grand View Mining & Development Co., of Twin Bridges, Mont., is situated near the crest of the Tobacco Root Range, about 11 miles east of Twin Bridges, the nearest station on a branch of the Northern Pacific Railroad from Whitehall. It is situated near the summit of Little Baldy Mountain and is reached by wagon road up Goodrich Gulch from Twin Bridges.

The deposit consists of a vein of porous and well-crystallized cerussite heavily impregnated with iron oxide and containing perhaps  $\frac{1}{2}$  to 1 per cent of well-crystallized wulfenite. The ore is reported to carry 30 to 50 per cent of lead, 2 to 6 ounces of silver, and 0.2 ounce to 1.5 ounces of gold per ton.

At the time of the author's visit to the property in the late fall of 1913 little development work had been done, and the outcrop of the deposit was covered with several feet of snow, so little exact information as to the size and extent of the deposit could be obtained. A short tunnel had been driven only a few feet below the outcrop, and intersected the vein at a point where it was 17 feet horizontally be-

---

<sup>a</sup> Letter of Dec. 9, 1914, from F. W. Hall, Wise River, Mont.



tween walls, and a few feet of drifting had been done on the vein at this point. The strike of the vein as indicated by this work was practically east, and the dip appeared to the author to be about  $35^{\circ}$  to  $40^{\circ}$  to the north. However, the management of the property seemingly has a radically different idea as to the dip of the deposit, because the company has started a tunnel several hundred feet below, on the south side of the mountain, and is driving it north to intersect the vein, whereas if the deposit dips in the direction the author supposes the vein could be much more readily reached by a tunnel from the north side of the mountain. In the upper workings black limestone seems to form both the footwall and the hanging wall of the deposit.

A microscopic examination of a large dark-colored wulfenite crystal, said to have come from this deposit and given to the author by Henry Schmidt, president of the company, showed the presence of metallic gold in the crystal, the occurrence being seemingly similar to that of the native metal in wulfenite crystals from the Old Yuma mine near Tucson, Ariz. (see p. 115).

Because of the small amount of development work on the property, it is impossible to say whether the occurrence of wulfenite in the ore may prove of commercial importance as a source of molybdenum. However, if concentration of the ore is attempted with a view to saving the wulfenite, great difficulty is almost certain to be encountered in separating it from the cerussite, which is nearly of the same specific gravity.

#### NEW MEXICO.

Few occurrences of molybdenite have been reported in New Mexico, and only one deposit in the State, that of the Romero Mining Co., near Porvenir, San Miguel County, has been visited by the author. Other occurrences are at Rociada, about 6 miles north of Porvenir, in the same county; at Bromide, in Rio Arriba County; and at the Modoc mine, in the Fierro district, Grant County. In all these localities the molybdenite is associated with copper ores.<sup>a</sup>

The mineral also occurs in quartz breccia in a deposit near Magdalena, Socorro County,<sup>b</sup> and associated with lead and silver ores at Organ, Dona Ana County.<sup>c</sup>

Deposits of wulfenite are confined almost entirely to Dona Ana, Sierra, and Grant Counties, in the southwestern corner of the State. The mineral occurs rather commonly with silver ores in the Organ district, in Dona Ana County, and many well-crystallized specimens have been obtained from the Bennett mine at Organ. Most of these specimens show beautiful yellow crystals of wulfenite associated with

<sup>a</sup> Sanford, Samuel, and Stone, R. W., *Useful minerals of the United States*: U. S. Geol. Survey Bull. 585, 1914, p. 132.

<sup>b</sup> Specimens in collection of Bureau of Mines.

<sup>c</sup> Sanford, Samuel, and Stone, R. W., *loc. cit.*

cellular quartz and secondary calcite and limonite. In Grant County the mineral occurs in deposits near Lordsburg<sup>a</sup> and also in the Lucky Bill mine near Santa Rita, but as far as known these occurrences are only of mineralogical interest. Plate IV, *B*, shows a group of wulfenite crystals of unusual form from the Lucky Bill mine. In Sierra County wulfenite has been mined at Hillsboro,<sup>b</sup> and a small production of the mineral was derived by Mr. Ralph Widener in 1914 from the Gladys claim, in the Caballos Mountains,<sup>c</sup> near Cutter. The occurrence of the mineral has also been reported from the Belcher mine, in Santa Fe County.<sup>d</sup>

#### MOLYBDENITE PROPERTY OF THE ROMERO MINING CO.

The Romero Mining Co., of Las Vegas, has done considerable development work on a molybdenite prospect situated about a mile and a half from the Porvenir Hotel at Porvenir, San Miguel County. This town is about 18 miles northwest of Las Vegas, and may be reached from there over a good wagon road or over a part of the same road from the northerly terminal of the Hot Springs Railroad, from which it is about 8 miles distant.

The vein material of this deposit is seemingly a pegmatite, consisting largely of quartz and feldspar, with smaller amounts of biotite and occasionally fluorite. The molybdenite, which is closely associated with chalcopyrite, occurs in masses up to 2 or 3 inches in maximum diameter in occasional pockets in the vein material. The ore also contains scheelite, bismuthinite, malachite, molybdite, and various iron oxides. Plate XVI, *B*, illustrates a cross section of a specimen of molybdenite ore from this property. The interesting ore structure shown, although not typical of the ore as a whole, was noted in a number of specimens. The country rock is a fine-grained pink feldspathic granite, which upon microscopic examination shows the following minerals: Quartz, orthoclase, microcline (very abundant), albite, a small amount of biotite, and a few minor accessory minerals.

When the author visited this property, in the winter of 1913, it was covered with snow and the outcrop of the deposit could not be examined. At that time development work consisted of a tunnel about 700 feet long and a shaft said to be about 75 feet deep. The bottom of this shaft was filled for 20 feet or more with material that had caved from the sides. Twenty or thirty tons of rich molybdenite ore associated with tungsten, bismuth, and copper minerals was reported to have been taken from several pockets in this shaft, but at the time of the author's visit there was no ore visible either in the

<sup>a</sup> Sanford, Samuel, and Stone, R. W., *op. cit.*, p. 134.

<sup>b</sup> Sanford, Samuel, and Stone, R. W., *loc. cit.*

<sup>c</sup> Hess, Frank L., The production of cobalt, molybdenum, nickel, etc.: Mineral Resources of U. S. for 1914, U. S. Geol. Survey, 1915, p. 925.

<sup>d</sup> Specimen 84293 in U. S. National Museum.

shaft or in the tunnel, which was being driven with the expectation of crosscutting the ore-bearing pegmatite at depth. A considerable part of the rich ore taken from the shaft was sacked and stored in a small experimental mill that had been erected near the property. A part of it had been treated by crushing, followed by screening and hand picking to remove the coarser pieces of molybdenite, followed by concentration on a Wilfley table. The author was shown several small lots of coarse high-grade molybdenite and several tons of concentrates containing about 7 per cent  $\text{MoS}_2$  and rich in copper, which was obtained by this method of treatment. It was stated that tests to recover the molybdenite and copper by means of an oil-flotation process tried on an experimental scale at the mill had resulted in failure. It is understood that a few hundred pounds of coarse high-grade molybdenite, which presumably was recovered from some of the sacked ore in storage at the time of the author's visit, was sold by the company in 1914.

### WASHINGTON.

#### CROWN POINT MINE.

The Crown Point mine of the Aurelia Crown Co. of Seattle is situated in the Chelan National Forest, in the NW.  $\frac{1}{4}$  T. 31 N., R. 16 E., at the head of Railroad Creek, Chelan County. It is near the summit of the Cascade Range at an elevation of over 5,000 feet. (See accompanying sketch map, fig. 1.) The property is reached from Chelan Falls, on the Great Northern Railway, via stage to Chelan or Lakeside, a distance of 3 or 4 miles, thence by steamer up Lake Chelan for about 40 miles to Lucerne, and then by wagon road and trail up Railroad Creek for about 18 miles. The first 3 or 4 miles of this last part of the journey are over a poor wagon road which ascends the hill back of Lucerne with many steep grades and sharp curves to connect with an abandoned railroad grade which continues up the north bank of the creek to a point about 12 miles from its mouth. From the up-stream end of the railroad grade a poor trail leads to the property, about 6 miles distant. The elevation of Lake Chelan at Lucerne is 1,072 feet. Accordingly the vertical rise from the lake to the mine is about 4,000 feet, and the average grade of the road and trail for the entire 18 miles is over 4 per cent.

The molybdenite-bearing deposit at the Crown Point mine consists in the main of a nearly horizontal vein of white vitreous quartz, which outcrops for a distance of several hundred feet on the north-east face of an almost perpendicular granite cliff at a height of 700 or 800 feet above its base. This vein dips slightly (perhaps  $5^\circ$  to  $6^\circ$ ) west. On the outcrop its maximum thickness is about 3 feet, and it pinches gradually to 3 or 4 inches at both ends of the outcrop. The vein also decreases rapidly in thickness as it runs back into the cliff, and



in a distance of 75 to 100 feet it practically pinches out. Plate XVII, A, shows the outcrop at the mouth of one of the short tunnels by which the property is developed. Attention is called to the pinch of the vein toward the left as it runs into the cliff and to the well-defined footwall and hanging wall. The old tree stub at the right leans out from the face of the cliff, which drops almost perpendicularly from the mouth of the tunnel for 600 or 700 feet.

An inspection of the local topography makes it evident that the hill in which the vein occurs once extended much farther to the northwest, and that a considerable part of it has been removed by erosion to form the valley on which the present cliff faces. It is certain

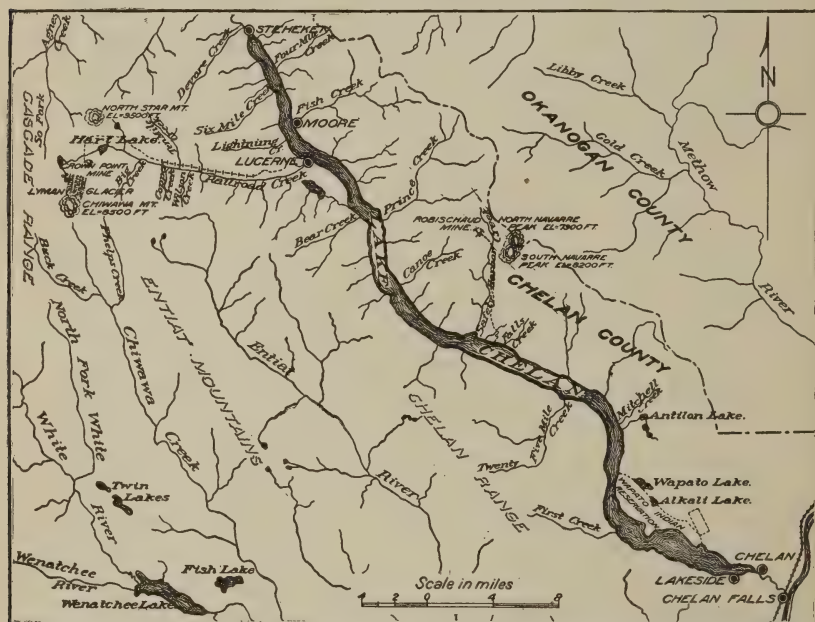


FIGURE 1.—Sketch map of the vicinity of Lake Chelan, Wash., and vicinity, showing the situation of the Crown Point and Robischand mines.

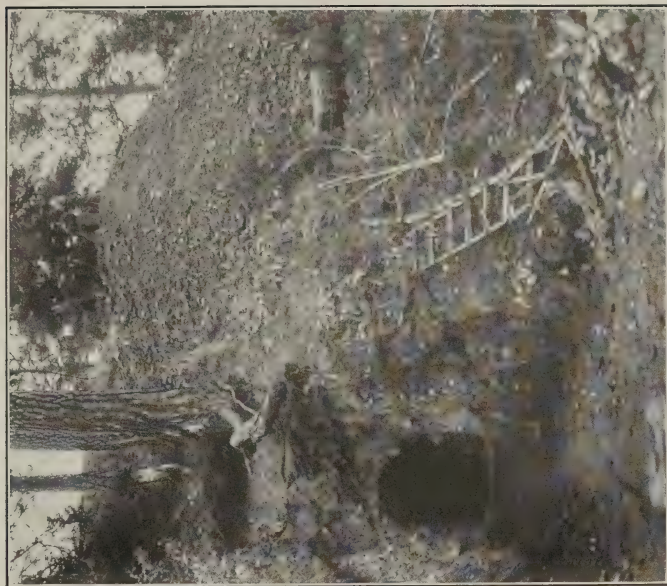
that the molybdenite-bearing vein extended into this eroded part, but there is no evidence from which to determine how much of the vein has been carried away.

The molybdenite occurs scattered irregularly through the quartz in masses ranging in size from minute specks to pieces 4 to 5 inches in maximum diameter. In general the molybdenite is unusually well crystallized, and the deposit probably affords finer specimens of the mineral than are to be obtained from any other known occurrence. Most of the crystals have the form of flat, hexagonal pyramids, the majority of which are beautifully striated. Some of them are one-half to 1 inch high, and 4 to 5 inches in maximum diameter, and weigh one-fourth to one-third of a pound each. The contrast afforded





A. OUTCROP OF UPPER VEIN AT MOUTH OF TUNNEL NO. 1,  
CROWN POINT MINE, RAILROAD CREEK, CHELAN COUNTY,  
WASH.



B. LOWER TUNNEL AND DUMP FROM UPPER WORKINGS, STARR  
MINE, NEAR LOOMIS, OKANOGAN COUNTY, WASH.

LIBRARY  
OF THE  
UNIVERSITY OF ILLINOIS

by these lustrous black crystals on a background of clean white quartz is strikingly beautiful. Photographs of some of these crystals and of a characteristic ore from this deposit are reproduced in Plates I and III, A. A few masses of molybdenite larger than those mentioned above are found, and the company states that "some have been mined that weighed over 5 pounds," and that "one weighed 11 pounds." In general, the molybdenite does not occur in the center of the vein, but is confined to horizons extending from a line 2 to 3 inches from the center of the vein to a line the same distance of either wall. The accessory minerals are unimportant. Small quantities of chalcopyrite and copper carbonates derived from it occur at several places in the vein, also occasional small patches of galena and sphalerite. These minerals are, however, largely confined to occurrences along the walls or in the immediately adjoining country rock. On the whole, the quartz-vein material is remarkably free from all minerals except molybdenite. In several places along the outcrop and in the tunnels where molybdenite is exposed to the air it has altered to molybdite, and fine specimens of partly altered crystals may be obtained.

The country rock is a medium to fine grained, greenish-gray biotite granite in which the feldspars are kaolinized to a considerable extent. The biotite is also more or less altered and is in various stages of chloritization. According to analyses made by Zaumbrecker and Pierce,<sup>a</sup> of Northwestern University, the country rock does not contain molybdenum. Crook, in describing the occurrence of molybdenite at this mine,<sup>b</sup> cites the absence of molybdenum in the neighboring granite as an indication that the molybdenite in the vein has not been derived by lateral secretion.

About 10 feet immediately below the vein already described and roughly parallel with it there is a second smaller quartz vein which outcrops for a short distance along the face of the cliff. The vein is only a few inches in thickness, but it is said to be richer in molybdenite than the larger vein. Little development work has been done on it, and nothing can be said of its possibilities.

Development work on the upper vein consists of two short intersecting tunnels. The larger of these, locally known as Tunnel No. 1, runs about 200 feet west and is intersected at an acute angle by Tunnel No. 4, which is about 80 feet long and runs southwest. The author's examination of the property was of necessity confined almost entirely to an inspection of the upper vein as disclosed by these two tunnels, as a heavy snowstorm, in progress at the time of his visit, rendered work along the outcrop on the face of the cliff impossible.

---

<sup>a</sup> Crook, A. R., Molybdenite at Crown Point, Washington: Geol. Soc. of America Bull., vol. 15, 1904, p. 287.

<sup>b</sup> Crook, A. R., op. cit., pp. 283-288.

Equipment to serve the Crown Point and neighboring claims controlled by the same owners consists of a mill, machine shop, power house, sawmill, warehouse, three cabins, and a barn. The mill is situated on the slope at the foot of the cliff in which the molybdenite-bearing veins occur, and is connected with the tunnels above by an aerial tram. As the property had been closed for the winter at the time of the author's visit, entrance to the mill was not to be had, and no detailed description of its contents can be given here. The equipment is said to consist of crushing and concentrating machinery, the latter including a Schule dry concentrator imported from Hamburg, Germany, for treating the molybdenite ore.<sup>a</sup> The mill building also contains a small compressor plant operated by a Pelton wheel. The president and general manager of the company in speaking of the mill stated: "All the machinery works well except the Schule concentrator, which does not save the small particles of molybdenite; neither does it entirely separate all the quartz from the molybdenite we have concentrated."

There is a splendid supply of timber available for use in mining or building. About half of this is Douglas and white fir and the remainder consists of cedar, white and yellow pine, spruce, and hemlock. As already stated, the property lies within the boundaries of the Chelan National Forest: hence stumpage is for sale by the United States Government.

Much water power could be developed from a number of sources in the immediate neighborhood. Principal among these is Aurelia Lake, which has an area of about 80 acres, and is said to be capable of supplying water with a head of more than 1,600 feet within the confines of the property.

The mine is credited with a production of 10 to 12 tons of molybdenite ore in both 1901 and 1902.<sup>b</sup> Presumably this was high-grade material, although no statement regarding its quality is made. After leaving the property the author saw at the warehouse of the company at Lucerne about  $2\frac{1}{2}$  tons of high-grade mineral estimated to contain over 95 per cent  $\text{MoS}_2$  which was ready for shipment. This material was said to have been obtained by hand cobbing the ore and picking out the larger pieces of molybdenite.

The cost of mining at the property is considerable on account of the high cost of transporting supplies and the narrowness and slight dip of the ore body. The granite in which the vein occurs is hard and at least 3 or 4 feet of it must be mined. However, the vein stands up fairly well after the underlying granite has been removed, and with careful selective mining little sorting would be necessary, and the

---

<sup>a</sup> Editorial, Eng. and Min. Jour., vol. 78, 1904, p. 273.

<sup>b</sup> Pratt, J. H., Tungsten, molybdenum, uranium, and vanadium: Mineral Resources U. S. for 1902, U. S. Geol. Survey, 1903, p. 287.



little necessary would be rendered easy on account of the great dissimilarity between the vein material and the waste rock.

The precipitation during both summer and winter is great, and, owing to the altitude of the property, snow remains until early summer. The roads and trails from Lake Chelan to the property are generally open in June, and easy access to the mine may be had until the middle or last of September. On September 15, 1914, the company's men, who were carrying on development and assessment work, left the property for the winter, and on the next day, when the author visited the property, a heavy fall of snow took place.

If further development should disclose sufficient ore of similar character to that already in sight to warrant the installation of the necessary machinery, no difficulty should be experienced in concentrating the ore. A large part of the molybdenite could be efficiently recovered by hand picking after crushing the coarse pieces. The discard might then be reduced to perhaps 10 mesh and treated by electrostatic methods. Concentration tests of nearly similar ores lead the author to believe that a high recovery and a high-grade concentrate could be made.

#### **MOLYBDENITE NEAR LOOMIS, OKANOGAN COUNTY.**

There is a large deposit of low-grade molybdenite ore about 10 miles from Tonasket, Okanogan County, on one of the spurs of the Aeneas Mountains. It may be readily reached by automobile from Tonasket or from Loomis, both towns being about the same distance from the property. The deposit, which is owned by Andrew Starr, of Tonasket, appears to consist of a blow-out or chimney of medium-grained light-gray granite in which the molybdenite, associated with pyrite, occurs as small flakes up to an eighth of an inch in diameter. The writer would judge that the average molybdenite content of the granite was 0.5 per cent, but by careful hand picking ore containing 2 or 3 per cent of molybdenite might be obtained. At the time of the author's visit to the property in the late fall of 1914 the limits of the outcrop of the ore body had not been determined, as in most places the granite was covered with considerable soil, but so far as prospected it appeared to be about 200 by 400 feet. The principal development work consisted of two short prospect tunnels, the relative locations of which are shown in Plate XVII, *B*. The principal constituents of the granite are quartz, feldspar, and biotite. The feldspars have been considerably altered to sericite and kaolin. Accessory minerals are ilmenite, apatite, and zircon. Dr. F. B. Laney, of the United States Geological Survey, who kindly examined samples of ore from this deposit for the author, states that:

The molybdenite and a small amount of pyrite are distributed irregularly through the rock in such a manner as to suggest the possibility of their being original constit-

uents. In some places the molybdenite occurs in irregular fractures, but is by no means confined to them. Indeed it is distributed pretty well throughout the whole mass of the specimens studied, but appears to be more abundant in the more highly quartzose parts of the rock, thus possibly indicating that it is a late addition to the solidifying or partly solidified magma.

#### OTHER OCCURRENCES OF MOLYBDENITE IN WASHINGTON.

##### DEPOSIT NEAR SAFETY HARBOR CREEK, CHELAN COUNTY.

Molybdenite associated with chalcopyrite is reported to occur in Chelan County near the head of Safety Harbor Creek, which flows into Lake Chelan on its northwestern shore about 25 miles in an air line from the foot of the lake. The deposit is situated on the northwestern side of the creek, opposite the Navarre Peaks, and is at an elevation of about 6,000 feet. It is 10 miles over a good trail from the mouth of the creek to the camp. The situation of the deposit is shown in figure 1. The following notes regarding the property were kindly furnished <sup>a</sup> by the owner, P. Robischaud, of Lakeside, Wash. The deposit consists of two systems of veins, one running approximately east, the other north. About 700 feet of development work has been done, mostly in tunnels. The main tunnel is about 330 feet long, and 200 feet of drifts has been run from it. Twelve fissures running east and three running north have been cut. All of these are small, being 18 inches or less in width. The greatest depth reached on any of the veins is about 200 feet. The country rock is said to be porphyry.

Specimens of ore submitted to the bureau consist of a highly altered rock containing molybdenite and chalcopyrite. In most of the specimens the two minerals are not intimately associated, the molybdenite being confined to areas bounding those in which the chalcopyrite occurs; for example, in one piece showing a 4½-inch streak of chalcopyrite the molybdenite occurs in two bands about half an inch wide on either side of the chalcopyrite. The rock in which the sulphides occur has been so changed that determination of its original composition is impossible. The principal constituents are quartz, a carbonate that is seemingly siderite, and a phosphate that is probably apatite.

##### DEPOSIT ON SHEEP MOUNTAIN, OKANOGAN COUNTY.

J. M. Risley, of Twisp, Wash., has furnished the following information concerning a molybdenite deposit on Sheep Mountain in the northern part of Okanogan County.<sup>b</sup> The property, which consists of five claims, is situated about 1 mile south of Monument No. 11 of the United States-Canadian boundary line and about 20 miles from

<sup>a</sup> Letter of Nov. 21, 1914.

<sup>b</sup> Letters of Dec. 28, 1914, and July 23, 1915.

Ashnola Siding, a station in British Columbia on the branch line of the Great Northern Railroad to Princeton. The best route to the claim is said to be over a wagon road that runs from this station up the Ashnola River for 8 miles and thence by trail about 12 miles to the property. On the Washington side the deposit is about 55 miles by trail from Twisp.

The deposit consists of three roughly parallel veins 4 to 7 feet wide, the outcrops of which may be traced almost horizontally along the side of the mountain. The veins are well defined and traverse a granite country rock. The altitude of the lowest vein is about 8,000 feet and that of the highest is about 9,000 feet. The two lower veins are below timber line, but the third is above it. Development work consists of several open cuts and short tunnels. Specimens sent to the bureau show the vein material to be a pegmatite in which the molybdenite occurs as flakes up to a quarter of an inch in diameter, associated with molybdenite and pyrite. The specimens submitted were free from copper minerals and were estimated to contain about 1 per cent of molybdenite.

#### PROSPECTS NEAR OROVILLE, OKANOGAN COUNTY.

The author has noted the occurrence of small quantities of molybdenite associated with chalcopyrite in quartz in the Forty-ninth Parallel, Golden Chariot, and O. K. prospects, all of which are situated about 3 or 4 miles north of Oroville, Okanogan County, on the eastern slope of the hills on the west shore of Lake Osoyoos. The mineral also occurs near Concunully in the same county, associated with quartz, sericite, pyrrhotite, and secondary calcite.<sup>a</sup>

#### MISCELLANEOUS OCCURRENCES.

Other localities in the State in which molybdenite occurs are as follows: Near Skykomish, King County, with chalcopyrite and bornite;<sup>b</sup> in the Monte Cristo district, Snohomish County, as occasional fine flakes;<sup>c</sup> in Pierce County on the north side of Mount Ranier near the White River Glacier at an altitude of about 5,000 feet;<sup>d</sup> on Thunder Creek, a tributary near the head of the Skagett River;<sup>e</sup> in Ferry County on many prospects in the Sanpoil district;<sup>f</sup> also in the Metalline district in Stevens County.<sup>g</sup>

<sup>a</sup> Specimen in U. S. National Museum.

<sup>b</sup> Specimen in Museum of California Bureau of Mines.

<sup>c</sup> Spurr, J. E., The ore deposits of Monte Cristo, Wash.: U. S. Geol. Survey Twenty-second Ann. Rept., pt. 2, 1901, pp. 777-865.

<sup>d</sup> Letter from Milnor Roberts, University of Washington, Nov. 20, 1913.

<sup>e</sup> Idem.

<sup>f</sup> Sanford, Samuel, and Stone, R. W., Useful minerals of the United States: U. S. Geol. Survey Bull. 585, 1914, p. 205.

<sup>g</sup> Sanford, Samuel, and Stone, R. W., loc. cit.



# TABULATION OF MOLYBDENITE AND OF WULFENITE OCCURRENCES IN THE UNITED STATES.

Following are tabulations showing molybdenite and wulfenite occurrences in the United States, with remarks and references to descriptive literature.

## *Occurrences of molybdenite in the United States.*

State.	Remarks.	Reference.
Alaska:		
Jumbo mine, Sulzer.....	In altered limestone (?) with chalcopyrite.	U. S. Geological Survey Prof. Paper 89, 1915, p. 49.
Treadwell mine.....	Frequent in ore.....	Trans. Am. Inst. Min. Eng., vol. 35, 1905, p. 503.
Chilcoot Pass.....		Specimen in U. S. National Museum.
At mile 192 on Copper River & Northwestern R. R.		Min. and Eng. World, vol. 37, 1912, p. 582.
On Lost River, Seward Peninsula.	Fairly coarse aggregates in iron-stained siliceous gangue.	Specimens in collection of U. S. Geol. Survey.
Arizona:		
See pp. 52 to 58.....		
California:		
See pp. 58 to 63.....		
Colorado:		
See pp. 63 to 72.....		
Connecticut:		
At Haddam, Middlesex County.	In gneiss.....	Specimen in Brush collection, Yale University; and Dana, E. S., A system of mineralogy, 6th ed., 1911, p. 41.
At Saybrook.....		Dana, E. S., loc. cit.
South Glastonbury, Hartford County.	In pegmatite.....	U. S. Geol. Survey Bull. 585, 1914, p. 53.
Idaho:		
At Knob Hill, Kootenai County	On property of the Sauca Consolidated Co.	Min. Ind., vol. 16, p. 723.
Seven Devils district.....	In Chieftan Copper mine.....	Specimen in California Bureau of Mines Museum.
At Leadore, Lemhi County....	Small flakes with sericite.....	Specimens in collection of U. S. Geol. Survey.
On south fork of Salmon River, 14 miles from Warren.		Min. Ind., 1905, vol. 14, p. 450.
Maine:		
At Cooper, Washington County.	Flakes in pegmatite dikes and granite.	U. S. Geol. Survey Bull. 340, 1908, pp. 231-240; and U. S. Geol. Survey Bull. 260, 1905, pp. 197-199.
At Brunswick, Cumberland County.	With quartz, feldspar, and biotite.	Specimen in U. S. National Museum; U. S. Geol. Survey Bull. 340, 1908, p. 236.
At Blue Hill Bay and Camdage farm, Hancock County.	In large crystals.....	Dana, E. S., loc. cit.
At Bowdoinham, Sagadahoc County, and Sanford, York County.		Do.
At Tunk Pond, Hancock County.	Flakes in pegmatite dikes and granite.	U. S. Geol. Survey Bull. 340, 1908, pp. 231-240; Min. World, vol. 31, 1909, pp. 323, 324.
At Buckfield, Oxford County....		Eng. and Min. Jour., vol. 82, 1906, p. 1106; U. S. Geol. Survey Bull. 340, 1908, pp. 236-237.
At Greenwood, Oxford County.		Eng. and Min. Jour., vol. 82, 1906, p. 1106.
Mount Mica, 4 miles east of Paris.	In pegmatite.....	U. S. Geol. Survey Bull. 340, 1908, p. 237.
Massachusetts:		
At Shutesbury, east of Locke's pond.		Dana, E. S., loc. cit.
At Brimfield.....	With iolite.	Do.
At Rockport.....	In granite.	Letter from C. H. Warren, Boston, Mass.
At Quincy.....	do.	Do.
Minnesota:		
Near Portage, Aitkin County....		Min. Res. U. S., 1901, U. S. Geol. Survey, 1902, p. 267.
Montana:		
See pp. 72 to 76.....		



*Occurrences of molybdenite in the United States—Continued.*

State.	Remarks.	Reference.
Nevada:		
At Scott's camp, near Alpine.....		Letter from G. Scott, Alpine, Nev., Nov. 9, 1914.
Lander County.....		Specimen in Nevada exhibit of Pan.-Pac. International Expo., 1915.
Near Lewis, Lander County.....		Specimen in California Bureau of Mines Museum.
In Tule Canyon, Esmeralda County.	In quartz.....	Specimen in Nevada exhibit of Pan.-Pac. International Expo., 1915.
14 miles from Tonopah on Lone Mountain, Esmeralda County.		Eng. and Min. Jour., vol. 76, 1903, p. 667.
At Columbia, Esmeralda County.	With powellite, scheelite, and kaolin.	Specimen in U. S. National Museum.
10 miles west of Winnemucca, on Rose Creek, Humboldt County.		Eng. and Min. Jour., vol. 87, 1909, p. 775.
Winnemucca, Humboldt County.	With quartz, chalcopyrite, and gypsum.	Specimen in U. S. National Museum.
At Mena, Mineral County.....	Fine grains with molybdenite in siliceous gangue.	Specimens in collection of U. S. Geol. Survey.
26 miles north of Golconda, Humboldt County.	Fairly large aggregates in siliceous gangue.	Do.
Near Amos, Humboldt County.	Fine flakes with quartz sericite, and pyrite.	Do.
Near Yerrington, Lyon County.	Fairly large masses with quartz and feldspar.	Do.
Near head of Death Valley.....		U. S. Geol. Survey Bull. 585, 1914, p. 120.
In Yerington district, Lyon County.	In pegmatite.....	Do.
At Redlich, Mineral County....	In quartz veins.....	Do.
At Oak Springs, 45 miles north of Johnnie Siding, Nye County.	With powellite and scheelite.....	Do.
New Hampshire:		
At Westmoreland.....	In quartz and feldspar.....	Specimens in U. S. National Museum and Brush collection, Yale University; Dana, E. S., A system of mineralogy, 6th ed., 1911, p. 41.
At Llandaff.....	In tubular crystals.....	Dana, E. S., loc. cit.
At Franconia.....		Do.
Walpole.....	With kaolinized feldspar, muscovite, and molybdenite.	Specimen in U. S. National Museum.
Warren.....	With feldspar and hornblende....	Do.
New Jersey:		
At Stanhope, Sussex County...	In a fine-grained magnetite with massive pyrite in the Hude mine; also in Stanhope mine.	Eng. and Min. Jour., vol. 78, 1904, p. 101; U. S. Geol. Survey Bull. 585, 1914, p. 127.
McAfee.....	With augite and pyrrhotite in crystalline limestone.	Specimen in U. S. National Museum.
Hackettstown.....	With chalcopyrite and pyroxene..	Specimen in Field Museum of Natural History.
Edison, Sussex County.....	Crystals in granitoid rock in Ogden mine.	Specimen in American Museum of Natural History.
New Mexico:		
See pp. 77 to 79.....		
New York:		
In New Bed mines, Essex County.	With pyrrhotite.....	Trans. Am. Inst. Min. Eng., vol. 27, 1897, p. 199.
In Port Henry iron mines, Essex County.		Do.
Natural Bridge, Jefferson County.		Specimen in California Bureau of Mines.
Lyon Mountain, Clinton County		U. S. Geol. Survey Bull. 585, 1914, p. 138.
West Point, Orange County....		Do.
2 miles southeast of Warwick, Orange County.	With rutile, zircon, and pyrite..	Dana, E. S., A system of mineralogy, 6th ed., 1892, p. 41.
New York City, near Forty-second Street and First Avenue.	In gneiss.....	Specimen in American Museum of Natural History.
Tilly Foster mine, Brewster, Putnam County.	With hornblende, augite, chlorite, pyrrhotite, and magnetite.	Specimens in American Museum of Natural History and in Brush collection, Yale University.
Lake Champlain.....	With quartz, feldspar, and muscovite.	Specimen in U. S. National Museum.
Adirondack Mountains.....		Min. Ind., vol. 6, 1897, p. 485.

*Occurrences of molybdenite in the United States—Continued.*

State.	Remarks.	Reference.
North Carolina:		
Demming, McDowell County..	With quartz, muscovite-biotite schist.	Specimen in U. S. National Museum.
Near Pioneer Mills mine, Cabarrus County.	In granite and quartz.....	Genth, F. A., Minerals of North Carolina, 1891, p. 23.
Near Concord, Cabarrus County.	With pyrite in quartz.....	Dana, E. S., loc. cit.
Guilford County.....		Hintze, Carl, Handbuch der Mineralogie, vol. 1, 1898, p. 417.
Allegheny County.....	In Peach Bottom mine.....	Do.
At Haskett's, Macon County.....		Do.
West of the Blue Ridge.....		Do.
Oklahoma:		
Near Roosevelt, in Wichita Mountains.		Eng. and Min. Jour., vol. 77, 1904, p. 398.
Oregon:		
Copper Creek, Eagle Mountains, Baker County.	In white quartz.....	Bull. University of Oregon, vol. 1, No. 4, 1904, p. 100.
Nine miles from Huntington, down Snake River, Baker County.		Letter from J. L. McKeever, Chicago, Ill., Mar. 28, 1914.
Glacier Park, Middle Eagle Creek.		Bull. University of Oregon, vol. 1, No. 4, 1904, p. 100.
At Drum Lummon mines, in T. 6 S., R. 44 E.	With copper minerals in center of vein 8 feet wide.	Do.
Near Galice, Josephine County.		Min. Ind., vol. 16, 1907, p. 723, and vol. 18, 1909, p. 531.
Pennsylvania:		
Frankford, Philadelphia County	In gneiss, with feldspar, quartz, and biotite; also pegmatite; from Hoffman's quarry, with granular quartz and diopside.	Specimens in Brush collection, Yale University, American Museum of Natural History, and U. S. National Museum; Trans. Am. Inst. Min. Eng., vol. 31, 1901, p. 443.
Germantown.....	From stone quarries.....	Proc. of the Academy of Natural Science, 1887, p. 38.
Chester, near Reading.....	With feldspar, quartz, and muscovite.	Specimen in U. S. National Museum.
Morton, Delaware County.....	In Johnson's quarry, with feldspar, quartz, and biotite.	Do.
Avondale, Chester County.....	Leiper's quarry, with feldspar, quartz, and muscovite.	Do.
Rhode Island:		
Copper Mine Hill, Cumberland.	In serpentine matrix with malachite.	Specimen in American Museum of Natural History.
Near Sneece Pond, Cumberland.	In manganese ore.....	U. S. Geol. Survey Bull. 585, 1914, p. 165.
South Carolina:		
In Haile gold mine, northeast of Kershaw, on Lynches Creek, Lancaster County.	With pyrite carrying gold.....	U. S. Geol. Survey Bull. 293, 1906, p. 81.
Texas:		
South of Bluffton, Llano County.	In granite, with cyrtolite and ferugonite, with powellite.	Am. Jour. Sci., ser. 4, vol. 33, 1914, p. 485; specimen in U. S. National Museum.
Utah:		
Alta, Little Cottonwood district.	With quartz.....	Specimen in U. S. National Museum.
In Peruvian Gulch, 1½ miles west of Alta, Salt Lake County.	In siliceous gangue, with pyrite..	Specimens in collection of U. S. Geol. Survey.
Near Leamington, Millard County.	Flakes in quartz.....	Mineral Resources, U. S., 1901, U. S. Geol. Survey, 1902, p. 266.
Vermont:		
At Newport.....	With apatite.....	Dana, E. S., loc. cit.
Lexington.....	With feldspar and quartz.....	Specimen in U. S. National Museum.
Not given.....	With kaolinized feldspar and molybdite.	Do.
Washington:		
See pp. 79 to 85.....		
Wyoming:		
Albany County.....	With quartz.....	Field Museum of Natural History.
Encampment, Carbon County.		Specimen in California Bureau of Mines Museum.
The Illinois claim of Shoshone Mountain Mining Co., Kirwin.	Flakes in quartz.....	Letter from Henry Schnitzel, Deadwood, S. Dak., Apr. 3, 1914.

*Occurrences of wulfenite in the United States.*

State.	Remarks.	Reference.
Arizona: See pp. 45 to 52.		
California: Potrero, San Diego County.....	In quartz, with malachite and chalcopryrite.	Specimen in California Bureau of Mines Museum.
Lavis, San Bernardino County.....		Specimen in Museum of Yale University.
Inyo County.....	In Empire mine.....	Dana, E. S., A system of mineralogy, 6th ed., 1911, p. 991.
Mount St. Helena, Napa County.	In quartz.....	Specimen in California Bureau of Mines Museum.
Colorado: At Leadville.....	In ferruginous limestone.....	Specimen in Field Museum of Natural History.
Massachusetts: Southampton.....	With quartz and sometimes with pyromorphite.	Dana, E. S., loc. cit.; specimens in Field Museum of Natural History, American Museum of Natural History, and Brush collection, Yale University.
At Loudville, in Manhan lead mines, Hampshire County.	Sometimes hemimorphic crystals.	Dana, E. S., op. cit., p. 73, and U. S. Geol. Survey Bull. 585, 1914, p. 97.
Montana: See pp. 76 to 77.		
New Mexico: See pp. 77 to 78.		
Nevada: Eureka District, Eureka County, in Eureka Consolidated and K. K. mines; on Ruby Hill.	With quartz, calcite, and limonite. Possibly of commercial importance.	Trans. Am. Inst. Min. Eng., vol. 6, 1877-78, p. 559; U. S. Geol. Survey Bull. 585, 1914, p. 123; specimens in Brush Collection, Yale University, U. S. National Museum, Colorado State Museum, American Museum of Natural History, and in Field Museum of Natural History.
Tecoma, Elko County.....	White calcite on limonite.....	Specimen in American Museum of Natural History and Museum in California Bureau of Mines.
On Comstock Lode, Storey County.		Dana, E. S., op. cit., p. 990.
Redemption mine, Esmeralda County, $1\frac{1}{2}$ miles south of Hornsilver.		U. S. Geol. Survey Bull. 585, 1914, p. 123.
4 miles east of Crescent, Clark County.	With vanadinite.....	Do.
Quartette mine, Searchlight, Clark County.	With crysocola and cerussite....	Specimen in Harvard collection, Harvard University.
Eldorado mine, Belmont, Nye County.		Specimen in Brush collection, Yale University.
New York: Near Sing Sing.....	With vanadinite and pyromorphite on crystalline limestone.	Dana, E. S., op. cit., p. 991.
Pennsylvania: Phoenixville, Chester County..	With granite, pyromorphite, anglesite, cerussite, quartz, and galena.	Dana, E. S., loc. cit.; specimens in Brush collection, Yale University; American Museum of Natural History, U. S. National Museum, and Field Museum of Natural History.
Perkiomen, Montgomery County.	With pyromorphite in quartz....	U. S. Geol. Survey Bull. 585, 1914, p. 164; specimens in Brush collection, Yale University, and in American Museum of Natural History.
Eaton mines, near Audubon, Montgomery County.		U. S. Geol. Survey Bull. 585, 1914, p. 164.
Pequea mines, Lancaster County.		Do.
South Dakota: Richmond mine, Black Hills..	With quartz, cerussite, and limonite.	Specimen in U. S. National Museum.
Utah: At Alta in the Continental-Alta and City Rocks mines, Little Cottonwood district, Salt Lake County.	In limestone with lead ore.....	U. S. Geol. Survey Bull. 340, 1907, pp. 239-240.
Lucien district, Box Elder County.	With limestone.....	Specimen in Field Museum of Natural History and Brush collection, Yale University.

*Occurrences of wulfenite in the United States—Continued.*

State.	Remarks.	Reference.
Utah—Continued.		
In Tecamah mine.....		Dana, E. S., loc. cit.; specimen in Brush collection, Yale University.
In Harrington Hickory mine, Beaver County.		Specimen in collection of U. S. Geol. Survey.



### PART III. CONCENTRATION.

#### IMPORTANCE OF SUCCESSFUL CONCENTRATION OF LOW-GRADE ORES.

Successful concentration of the low-grade ores of molybdenum is the one factor on which the future of molybdenum mining depends more than on anything else. Outside of a small and irregular production of wulfenite concentrates, the molybdenum markets of the world have to date been supplied largely by a molybdenite product containing 50 to 90 per cent  $\text{MoS}_2$  obtained by the primitive process of cobbing and hand picking high-grade ore. In a few instances hand picking has been supplemented by coarse crushing, preferably in rolls or some similar type of machine that tends to flatten the crystals or flakes of molybdenite, followed by screening to separate the coarser particles of molybdenite from the gangue. That such a method is extremely wasteful is self-evident, and probably in every instance where it has been employed more molybdenite has been thrown on the dump as waste than has been saved. But aside from the inefficiency of the process, the known supply of rich ore on which it can be applied with success, even at the present high price of the molybdenite product, is so small as to promise a yield of only a few tons. Barring the discovery of extensive high-grade deposits of molybdenite, the product obtained by cobbing, hand picking, and screening out the coarser flakes of the mineral will not yield a hundredth part of the tonnage of concentrates requisite to the proper development of the market. Neither will the few known commercial deposits of wulfenite, which is easily concentrated, be able to supply the demand. Low-grade molybdenite ores, of which there are large deposits containing 1 to 3 per cent  $\text{MoS}_2$ , must therefore be treated.

As yet it has not been proven that any of the chemical or metallurgical processes so far used or proposed for extracting molybdenum are nearly as economical as mechanical processes of concentration when directly applied to low-grade molybdenite ores, particularly if the deposits are situated at a distance from railroad transportation. It appears, therefore, that the mechanical concentration of such ores, including a study of methods for the removal of deleterious elements, such as copper, from the concentrates is the chief problem to be considered. The Bureau of Mines hopes by investigation of these questions to obtain results that will point the way to the establishment of a successful industry.

In this report the question is necessarily taken up in a preliminary way as the bureau has at present no equipment of its own for conducting concentration tests. However, a discussion of the more successful existing methods of concentration is attempted here, and is supplemented by the results of a number of concentration tests of ores selected as representative of typical deposits. The tests described were made possible through the kindness of various companies and individuals who placed the equipment of their testing laboratories at the author's disposal. These tests were of necessity made on laboratory-size machines and on small lots of ore, and the results must be considered only as indicative of what may be expected from the treatment of commercial quantities of ore on machines of standard size.

With proper equipment at its command the bureau hopes to develop improved processes of concentration of these low-grade ores, the results of its work to appear in a succeeding report.

Of the two commercial ores of molybdenum, namely, molybdenite and wulfenite ores, the first are not amenable to the ordinary processes of concentration by jigs, tables, vanners, slimers, etc., such as are usually employed in treating ores of copper, lead, zinc, gold, and silver, as the grains or flakes of molybdenite even when of considerable size float readily on water and therefore are lost if these methods are employed. On the other hand, little difficulty is experienced in the adaptation of the ordinary jig and table processes to wulfenite ores, as they are readily wetted, and because of the high specific gravity (6.7 to 7) of wulfenite it is much heavier than any of the gangue-forming constituents of most of its ores. On account of the wide difference in the character of molybdenite and of wulfenite ores and the necessity of subjecting them to entirely different processes of treatment, the concentration of each is considered separately.

### CONCENTRATION OF MOLYBDENITE ORES.

Until recently, practically all the processes of concentration of molybdenite that have been attempted on a commercial scale in the United States have been unqualified failures. A number of companies have erected mills and installed various rolling and screening processes, pneumatic separators, and wet concentrators similar to or adaptations of those usually employed in the treatment of lead, zinc, and copper ores. These mills have operated only a few weeks at the most, and, as far as the author knows, only one company produced more than a ton of marketable concentrates. In several instances, the processes were absolutely unsuited to the ores to which they were applied, and their unsuitability could have been conclusively shown by a trial on an experimental scale. In other instances processes that

seemed to promise success from small-scale tests were failures on a commercial scale owing to difficulties in their practical application.

Notwithstanding this record of continuous failures, and although the concentration of molybdenite ore has been considered one of the most difficult problems in ore dressing, there are methods of treatment that have been developed for some time that can be applied to many molybdenite ores with certainty of success. Before proceeding to describe these processes it may be well to state that there is no method suitable for the treatment of molybdenite ores in general, but that each particular ore usually involves its own individual concentration problems; a method or combination of methods that might give highly satisfactory results with one ore might prove an unqualified failure with another. In describing each process the author will attempt to show to which particular type of ores it is adapted, and how some ores are best treated by a combination of processes.

Methods of concentration that have been used with success with molybdenite ores may be grouped under three general heads, as follows: (1) Rolling and screening processes; (2) electrostatic methods; (3) flotation processes. Each of these is considered separately.

#### ROLLING AND SCREENING PROCESSES.

A considerable proportion of the molybdenite in some ores in which the mineral occurs as large flakes or crystal aggregates may be recovered by crushing the ore in rolls and then screening the crushed material. The success of the process depends entirely upon flattening the molybdenite masses into flakes the maximum diameters of which are in excess of those of the particles of gangue material. Usually before the ore is screened it is passed through a series of rolls to break the gangue into small particles. The material passing through the first screen is again crushed in rolls and screened, and the process repeated as many times as may appear to be advantageous.

About 10 years ago the American Molybdenum Co., of Boston, erected a mill at Cooper, Me., for the recovery of molybdenite by the process outlined above. Hess <sup>a</sup> comments on this plant as follows:

The plant consisted of a 35-horsepower boiler and engine, a Sturtevant jaw crusher and roll, and four sets of special rolls, each 3 feet in diameter and 10 inches wide. The crusher was but a couple of feet above the floor, from which the material, crushed to about one-fourth inch square, was elevated to the Sturtevant roll, 18 inches in diameter by 4 inches wide, which reduced the ore to about one-eighth inch. It was then elevated to a bin at the top of the building, from which it fell to a series of two special rolls, thence was elevated to a third special roll, and run through a 34-mesh screen. The molybdenite caught on the screen was delivered to a box at the end. The material going through the screen was carried by an elevator and screw conveyor to a fourth roll, from which it fell onto a 40-mesh screen and from that to a 60-mesh screen. What

<sup>a</sup> Hess, F. L., Some molybdenum deposits of Maine, Utah, and California: U. S. Geol. Survey Bull. 340, 1907, p. 233.



went through the 60-mesh screen was elevated and sent to the tailings pile. It is readily seen that the repeated elevations of the material meant a considerable waste of power. The mill ran only six weeks, and is said to have made about a ton of concentrates, a portion of which seen by the writer was very clean. In the tailings some fine flakes of molybdenite were found, but the amount seemed small. Such a process, if mechanically perfected, might work profitably on deposits where, as in this one, the molybdenite flakes are comparatively broad, but would be wholly unsuited to deposits like many of those in Colorado and elsewhere, in which the individual flakes are of almost microscopic size.

Modifications of the rolling and screening process have been employed at other molybdenite mines both in this country and in Australia. All of them have proved inefficient, and in most instances where they have been used as the sole method of recovery more of the mineral has been thrown on the waste dump than has been saved. In the author's opinion, rolling and screening are of value only as preliminary treatment to remove the coarser and heavier masses of molybdenite from ores in which the mineral occurs in fairly large aggregates. The application of such treatment to this type of ore may be desirable in a few instances, as it saves a considerable part of the molybdenite in coarse form, thereby doing away with losses that would inevitably occur if the large pieces were broken up. How far rolling and screening should be continued depends, of course, on the nature of the ore and on what process of treatment is to follow. In general, however, screening to recover a concentrate should not proceed beyond the point where the succeeding process will begin to handle the material efficiently. Ores containing mica are of course unsuited to this method.

## ELECTROSTATIC PROCESSES.

### PRINCIPLES OF SEPARATION.

Molybdenite, in common with many other minerals, such as chalcopyrite, pyrite, pyrrhotite, and galena, is a good conductor of electricity as compared with quartz, feldspar, and most other silicates, calcite, etc., which are poor conductors. Hence it may be separated from a majority of the gangue minerals with which it occurs by electrostatic processes. The principle on which these processes are based is that two bodies charged with electricity of similar sign repel each other, but when charged with electricity of opposite signs they attract each other. When minerals that are good conductors of electricity come in contact with a charged body or an electrical field, they almost instantly attain an electrical equilibrium with that body or field and are repelled, whereas as regards poor conductors the time necessary for equilibrium to be reached is so appreciable that if the contact is of short duration the repelling force is negligible. Further, if the mixture of two minerals that are good and poor con-



ductors—for example, molybdenite and quartz—is charged to a high potential with electricity of one sign and placed in contact with a body charged with electricity of opposite sign, the mineral that is a good conductor (molybdenite) almost instantly assumes the charge of the body with which it is in contact, and is repelled. On the other hand, the mineral that is a poor conductor (quartz) is only slowly affected and retains its own charge of opposite sign for an appreciable length of time, and in consequence is attracted to the body with which it is in contact.

#### RANGE OF APPLICATION.

Theoretically a separation can be effected between any two minerals that differ in degree of conductivity, regardless of whether they are good or poor conductors, but in commercial work it has been impossible to make a separation of good conductors, as there is practically no difference in the time required for them to reach an electrical equilibrium. Therefore, although molybdenite may be separated from quartz, limestone, porphyry, granite, etc., it can not be separated electrostatically from chalcopyrite, pyrite, pyrrhotite, etc. The table following lists the more common minerals and rocks with which molybdenite occurs and shows whether they are good or poor conductors of electricity. In other words, the table indicates whether the mineral mentioned can or can not be separated from molybdenite by electrostatic means. However, it does this in only a general way, as many minerals of varying composition, such as garnet and sphalerite, are sometimes conductors and sometimes nonconductors. Accordingly it is unsafe to predict just what separations may be effected, and the suitability of electrostatic processes for the treatment of any molybdenite ore can be positively ascertained only by actual tests.

*Minerals and rocks associated with molybdenite, showing whether they are good or poor conductors of electricity.*

Good conductors.	Poor conductors.
Chalcopyrite.	Quartz.
Bornite.	Feldspar.
Chalcocite.	Granite.
Most other copper minerals.	Quartzite.
Pyrite.	Porphyry.
Pyrrhotite.	Garnet. <sup>a</sup>
Magnetite.	Most silicates.
Most other iron minerals.	Calcite.
Galena.	Limestone.
Hornblende.	Sphalerite. <sup>a</sup>
Garnet. <sup>a</sup>	
Sphalerite. <sup>a</sup>	

<sup>a</sup> Some varieties are good conductors; others are poor conductors.

For the successful application of electrostatic methods to the concentration of molybdenite the ore must be one in which the individual particles of molybdenite are of fair size and in which the gangue and other principal associated minerals are nonconductors. Ores in which the molybdenite is extremely fine to almost amorphous requires fine crushing to free the individual grains from the gangue. Such crushing is not desirable, as it makes dust that usually interferes with the separation and also lowers the capacity of the separators. Thus, the best results are obtained from the treatment of coarse material, so that the ores that are best suited for electrostatic separation are those in which the particles of molybdenite are flaky and of good size, and in which the gangue is clean quartz, granite, etc.

The main requisite in the treatment of the ore is that it be perfectly dry and warm. The necessity of removing all moisture can not be emphasized too strongly, as its presence, even in small quantities, in quartz, feldspar, etc., make these minerals conductors. Ore as coarse as 6-mesh and as fine as 200-mesh may be treated successfully, but, as already stated, the best results are usually obtained with coarse material. Close sizing of the ore is generally unnecessary, but it is apt to improve results, especially in the treatment of coarse material. Dust in the ore should be removed by a blower or other device before the ore is treated.

#### DESCRIPTION OF SEPARATORS.

The electrostatic separators used in this country are of two main types. One of them consists essentially of a series of metal rods or electrodes charged to a high potential and placed one above the other in an insulated frame. On the top of this frame is a hopper from which the ore is fed by a revolving roll. The ore falls from the roll against the first electrode, and good conductors that come in contact with it are repelled. The residue is thrown against the second electrode by means of a deflector, and more mineral particles that are good conductors are thrown out. The operation is repeated any number of times desired simply by increasing the number of charged rods or electrodes. Generally a standard separating unit of this type consists of a "rougher" and two "cleaners," each having six electrodes. The cleaners are placed back to back and the rougher is placed above them in such a manner that the concentrates from the rougher fall against the first electrode of one cleaner, and the tailings against the first electrode of the other cleaner. In this way the rough concentrates and tailings are re-treated separately. With the exception of the feed roll there is no moving part in the unit, and, consequently, its operation is extremely simple, and the cost of upkeep negligible. High-voltage electricity for charging the electrodes is supplied by stepping up current of ordinary voltage by

means of a rotary transformer. Thus the operation of the separator is independent of weather conditions, which so noticeably affect the operation of electrostatic generators.

The other type of separator has a revolving metal electrode onto which ore is fed. Opposite and adjacent to it is another electrode consisting of a row of sharp metal points. The first electrode is grounded, and the second is connected to one pole of an electrostatic generator, the other pole of which is grounded. The electricity delivered to the pointed electrode is discharged by convection to the grounded electrode and is dissipated as fast as received. The revolution of the grounded electrode carries the mineral particles through the convective discharge and results in their receiving a charge. Good conductors immediately lose this charge to the grounded electrode on which they rest; consequently they are neither attracted nor repelled by the electrode but simply fall off of it as it revolves. On the other hand, mineral particles that are poor conductors accumulate a charge on their surfaces nearest the pointed electrode, and on account of their nonconducting character they are polarized and are caused to adhere to the revolving electrode. These particles are brushed off the back of the electrode by a felt roller.

Three of the separating units described above are generally combined in one machine for commercial work in such a manner as to provide for three successive treatments of the mineral particles that are good conductors. In the separation of molybdenite and quartz, for example, a rough concentrate would be made on the first unit and it would be further cleaned on the second and third units.

This type of separator is open to the objection that it is dependent on electrostatic generators for its supply of electricity, and in moist climates or wet weather these give much trouble in their operation.

#### CONCENTRATION TESTS.

The tests described below were made on molybdenite ore submitted by the Santa Maria Molybdenum Mining & Milling Co., of San Diego, Cal. The ore, which was said to be a representative sample from this company's property 40 miles east of that city, was an iron-stained granite containing about 0.5 per cent of molybdenite and a small quantity of pyrite. The molybdenite occurred in radiating masses from the size of a pea to an inch or more in diameter. Microscopic examination showed that the granite was composed of quartz, orthoclase, plagioclase varying from albite to oligoclase, biotite, and a few minor accessory minerals. This ore was selected for these tests as in a general way it was typical of ores from a large number of deposits.



On account of the low grade of the ore it would be necessary to hand pick it in a commercial operation, and therefore, previous to crushing for electrostatic separation, a hand-picking test was made of the sample submitted. The results of this test are given in the following table:

*Results of hand picking molybdenite ore from the Santa Maria Molybdenum Mining & Milling Co.'s property, 40 miles east of San Diego, Cal.*

Item,	Original ore.	Sorted ore.	Reject.
Weight, pounds.....	313.62	115.62	198.00
Per cent MoS <sub>2</sub> .....	<i>a</i> 47+	1.15	.08
Weight MoS <sub>2</sub> , pounds.....	1.488	1.330	.158
Extraction MoS <sub>2</sub> , per cent.....		89.38	
Loss MoS <sub>2</sub> , per cent.....			10.62

*a* Calculated.

The lot of ore, which weighed approximately 313 pounds, was cobbled down to pieces 2 inches in maximum diameter, and hand picked. One hundred and ninety-eight pounds of the material was discarded and the remaining 115 pounds of selected ore was crushed in a laboratory-size jaw crusher to pass a 10-mesh screen. The discarded ore was also crushed through a 10-mesh screen, and both products were sampled and assayed. They contained 1.15 and 0.08 per cent MoS<sub>2</sub>. The ratio of concentration obtained was approximately 1 to 3, and the loss of molybdenite in the discarded material about 10 per cent.

For the electrostatic tests the hand-picked ore, which had been crushed through a 10-mesh screen, was screened into three sizes—through 10-mesh on 20-mesh, through 20-mesh on 60-mesh, and through 60-mesh. The coarser size was treated on a Sutton, Steele & Steele type of concentrator and the two finer sizes on a concentrator of the Huff type. The results of these tests are given in the following table:

*Results of electrostatic-concentration tests of hand-picked molybdenite ore from the Santa Maria Molybdenum Mining & Milling Co.'s property, 40 miles east of San Diego, Cal.*

THROUGH 10-MESH ON 20-MESH SUTTON, STEELE & STEELE TYPE CONCENTRATOR.

Item.	Ore.	Concentrates.	Middlings.	Tailings.
Weight, pounds.....	30.35	0.48		29.87
Per cent MoS <sub>2</sub> .....	<i>a</i> 1.58	87.17		.21
Weight MoS <sub>2</sub> , pounds.....	.481	.418		.063
Extraction MoS <sub>2</sub> , per cent.....		86.90		
Loss MoS <sub>2</sub> , per cent.....				13.10

*a* Calculated.



Results of electrostatic-concentration tests of hand-picked molybdenite ore from the Santa Maria Molybdenum Mining & Milling Co.'s property, 40 miles east of San Diego, Cal.—Continued.

## THROUGH 20-MESH ON 60-MESH HUFF TYPE CONCENTRATOR.

Item.	Ore.	Concentrates.	Middlings.	Tailings.
Weight, pounds.....	41.49	0.52	0.10	40.87
Per cent MoS <sub>2</sub> .....	<sup>a</sup> 1.26+	76.66	41.87	.20
Weight MoS <sub>2</sub> , pounds.....	.523	.399	.042	.082
Extraction MoS <sub>2</sub> , per cent.....		76.29	8.03	
Loss MoS <sub>2</sub> , per cent.....				15.68

## THROUGH 60-MESH HUFF TYPE CONCENTRATOR.

Item.	Ore.	Concentrates.	Middlings.	Tailings.
Weight, pounds.....	37.18	0.36	0.45	36.37
Per cent MoS <sub>2</sub> .....	<sup>a</sup> .94+	46.78	23.63	.21
Weight MoS <sub>2</sub> , pounds.....	.350	.168	.106	.076
Extraction MoS <sub>2</sub> , per cent.....		48.00	30.29	
Loss MoS <sub>2</sub> , per cent.....				21.71

## SUMMARY.

Item.	Ore.	Concentrates.	Middlings.	Tailings.
Total weight, pounds.....	109.02 <sup>a</sup>	1.36	0.55	107.11
Total weight MoS <sub>2</sub> , pounds.....	1.354	.985	.148	.221
Per cent MoS <sub>2</sub> .....	<sup>a</sup> 1.24+	<sup>a</sup> 72.42+	<sup>a</sup> 26.90+	<sup>a</sup> .20+
Extraction MoS <sub>2</sub> , per cent.....		72.75	10.93	
Loss MoS <sub>2</sub> , per cent.....				16.32

<sup>a</sup> Calculated.

It will be noted that with the coarser material a concentrate containing 87.17 per cent MoS<sub>2</sub> was made, and that the recovery in these concentrates was 86.9 per cent. The concentrates from the next size contained 76.66 per cent MoS<sub>2</sub>, and accounted for 76.29 per cent of the molybdenite in the feed. However, a middling product containing 41.87 per cent MoS<sub>2</sub> was made, which increased the total extraction on this size of material to 84.32 per cent. The fine ore yielded concentrates and middlings that assayed 46.78 per cent and 23.63 per cent MoS<sub>2</sub>, and accounted for an extraction of 78.29 per cent. It will be noted that the best work was done on the coarse material, both as regards recovery and grade of concentrates.

A summation of the three tests shows that the average grade of the products made was as follows: Concentrates, 72.42 per cent; middlings, 26.9 per cent; and tailings, 0.20 per cent MoS<sub>2</sub>. The total recovery on the concentrates and middlings was 83.68 per cent. The results obtained were satisfactory when it is considered that the ore treated contained pyrite, which of necessity was concentrated with the molybdenite.

## SUGGESTED METHOD FOR TREATMENT OF A TYPICAL ORE.

For ores of the same general type as that mentioned above—that is, ores in which the individual particles of molybdenite are flaky and of fair size and the gangue is clean quartz, granite, or other non-

conductor—the following general outline of the proper method of treatment is suggested:

The original ore, or selected ore, if hand picking is advisable, should be crushed in a jaw or gyratory crusher to about one-half inch to 1-inch size and the product crushed in rolls to pass a 6-mesh screen. This material should then be treated in a direct-heat rotary drier until all moisture has been removed, and screened to sizes approximately as follows: Through 6-mesh on 12-mesh; through 12-mesh on 20-mesh; through 20-mesh on 40 or 60 mesh; through 40 or 60 mesh. If much dust is present in the finest size of material, its removal with some sort of blower or other device previous to electrostatic treatment will probably be advantageous. Each size of the dried material should be treated separately on whatever type of electrostatic concentrator has been selected. The best method of operating these separators depends, of course, on their type, and also on the character of the ore being treated. In the author's opinion it is generally advantageous to operate the concentrators so as to obtain as clean tailings as possible. In this way the bulk of the material is eliminated at once, and the re-treatment of the concentrates and of the middling product takes much less time than that required in re-treating the original tailings. This method of operation will generally yield a better grade of concentrates and a higher recovery than any other. Also, the grading of the concentrates and middlings so as to obtain the best prices for the combined products can be more readily accomplished by this method. In some instances the middling products from the treatment of the coarser sizes of ore consist largely of particles of gangue with included grains of molybdenite. Under such conditions, the middlings should be crushed, sized, and re-treated.

To the author's knowledge, there are no plants that are treating molybdenite ore commercially by electrostatic methods; hence, figures as to costs of operation are unavailable. It seems reasonable to infer, however, that they would be only slightly different from those for treating zinc or other ores to which electrostatic methods have been successfully applied. The costs of drying and separating zinc-iron concentrates by electrostatic methods, including only operating costs, do not usually exceed \$1 per ton of material treated; and the total costs, including operation, royalty, taxes, and all other overhead expenses, such as interest on investment and amortization charges, usually range from \$2.50 to \$3.50 per ton. The costs of treating molybdenite ore would probably be slightly in excess of these figures, as the capacity of the separators would, no doubt, be less than when operating on zinc-iron concentrates, where the weight of concentrates and tailings produced is more nearly equal than it would be with the usual molybdenite ores. Drying of the molybdenite ore would probably cost 10 to 20 cents per ton, depending on

the size and type of drier used, the cost of fuel and labor, and the percentage of moisture in the ore.

If the electrostatic separators were so arranged that the products could be taken care of automatically, one man could tend 8 to 10 concentrators. The other labor required for operating a mill capable of treating 50 tons of crude ore per day would probably not exceed four or five men per shift, and no high-priced labor would need to be employed on account of the simplicity of the process.

The author is confident that electrostatic processes will yield good results with all molybdenite ores to which they are suited and to which they are intelligently applied, and they are of particular value for treating ore that can not be treated by flotation methods on account of the absence of requisite water supplies.

### FLOTATION PROCESSES.

Molybdenite has the property, common in varying degree to most metallic sulphides, such as chalcopyrite, sphalerite, galena, pyrite, and pyrrhotite, of not being wetted readily by water, and, when dry and in small particles, of floating on a water surface. Moreover, like those sulphides, it is easily wetted by most oils. Further, in a pulp of crushed ore and water, oils have a preferential wetting action for particles of molybdenite as against particles of gangue minerals such as quartz, and this selective wetting action is decidedly increased if the water is slightly acidified. Particles of molybdenite so wetted with oil are covered with a buoyant water-repelling coating that materially assists their flotation. As the reasons for many of these phenomena—for example, the selective wetting action of oils and the increase of this selective action by acids—are not clearly understood, and as even an elementary discussion of the accepted theories of mineral flotation would be out of place here, the reader who wishes further information on the subject is referred to a clear and concise exposition by Hoover<sup>a</sup> and to an excellent paper by Mickle.<sup>b</sup> It suffices here to say that the phenomena mentioned above are the basis on which all flotation processes depend. In many processes the area of the effective surface of flotation is increased by the liberation of bubbles of gas or air in the liquid, the surface of each bubble acting in the same way as the horizontal surface of a liquid at rest. These bubbles may be of air and may be produced by violent agitation of the pulp or by release of air from solution in the liquid by a reduction of pressure, or they may be of carbonic-acid gas formed by the action of sulphuric acid on limestone or other carbonates or by other means.

<sup>a</sup> Hoover, T. J., *Concentrating ores by flotation*, 1912, 221 pp.

<sup>b</sup> Mickle, K. A., *Flotation of minerals*: Proc. Royal Soc. Victoria, vol. 23, pt. 2, 1911, pp. 555-585, abstracted by Eng. and Min. Jour., vol. 92, 1911, pp. 307-310, and vol. 94, 1912, pp. 71-76.



## WATER-FLOTATION PROCESSES.

## CHARACTER OF WATER FLOTATION.

Water flotation for the concentration of molybdenite depends solely on the fact that small dry particles of the mineral float readily on water, whereas the usual gangue material is easily wetted and sinks. They do not of necessity involve the use of oil, acid, or gas, and their application is extremely simple. The concentrators, which are of various types, consist essentially of a device for feeding the crushed ore in as thin a sheet and at as uniform a rate as possible onto a moving water surface in a tank and an arrangement, either by an overflow or a revolving belt of canvas or other suitable material, for discharging the floating film of concentrates into another tank. The tailings are usually drawn off from a spigot in the bottom of the first tank. In some types of apparatus the ore from the feeder is allowed to slide down an inclined plane or concave, over which a film of water is passed and from which the ore is discharged approximately in the plane of the water surface in the tank. In other apparatus the ore from the feeder falls on top of an almost submerged corrugated or canvas roller, the revolution of which carries the ore forward to the flotation surface. The object of all these devices is to place the ore on the water in a sheet only one mineral particle in depth with as little disturbance of the water surface as possible and with the majority of the particles of gangue already wetted.

Even with the best of feeding devices some particles of gangue fall on the floating film of concentrates or are otherwise mechanically entrained by it. Various methods of cleaning the film, such as allowing it to flow down an incline into a second tank, picking it up on a roller or belt and again discharging it to a flotation surface and dividing it into a large number of parts and agitating it by causing it to flow through the teeth of a comblike obstruction, are used.

No description of any particular water-flotation concentrator is attempted here, as detailed information both as to the design and the operation of a number of different types of machines is given in the references cited below.<sup>a</sup>

## NECESSARY FACTORS IN SUCCESSFUL WATER FLOTATION.

To be concentrated successfully by water flotation, a molybdenite ore should be such as to require only medium-fine crushing to liberate the molybdenite, and the gangue should be one in which the individual particles are readily wetted. Further, if a high-grade concentrate is to be obtained the ore must be practically free from other sulphides

---

<sup>a</sup> Wood, H. E., The Wood flotation process: *Trans. Am. Inst. Min. Eng.*, vol. 44, 1912, pp. 684-701; *Concentration of molybdenite ores: Eng. and Min. Jour.*, vol. 93, 1912, pp. 227-228.



such as pyrite, pyrrhotite, and chalcopyrite, which would be concentrated with the molybdenite. Proper treatment of the ore previous to flotation is of great importance. It should be reduced to approximately 10-mesh, or as much finer as may be necessary to liberate the mineral from the gangue, by crushing in rolls in such a manner as to make the quantity of fines as small as possible. Then if it is at all damp it should be thoroughly dried. Mechanical difficulties in the proper feeding of fine ore, and the great reduction in the capacity of the concentrators when fine material is treated exclusively, render advantageous in most instances the treating of the ore without sizing, although the capacity of the concentrators and the grade of concentrates made are considerably increased when coarse material that has been sized is treated.

The capacity of water-flotation machines varies with their type and with the character of the ore treated, and depends directly on the size to which the ore is crushed, the amount of fines made, the nature of the gangue, and the ratio of concentrates to tailings. In speaking of a particular type of concentrator, Wood <sup>a</sup> says:

A standard machine treating a 20-mesh quartz ore, using a 3-foot width of feed and having a 4-foot take-off belt, will vary in capacity from 1,000 to 2,000 pounds per hour, unless the ratio of concentration is low, in which case the capacity will be smaller. Some ores that possess an easily wetted gangue and call for a high concentration ratio can be fed rapidly at 20-mesh, 30-mesh, or 40-mesh. For instance, a 1 or 2 per cent molybdenite ore in a quartz gangue will give a clean concentrate, even if the ore is fed several times faster than an ordinary sulphide ore.

The writer thinks that the capacities stated above could be had only at a sacrifice of either the recovery or the grade of concentrates obtained, and that a machine with a feed 3 feet wide, handling 300 to 500 pounds per hour of ore crushed to pass a 20-mesh screen would be treating about the maximum quantity of material that it could separate efficiently. No figures as to cost of operation can be given, but they are presumably small as the concentrators require only moderate quantities of water and little power.

#### RESULTS OF TESTS WITH WOOD'S PROCESS.

The accompanying table shows the results obtained by Wood in 10 concentration tests of five different molybdenite ores. In every test except the last, in which the ore had been slightly roasted, the recoveries were good, averaging nearly 90 per cent. In general, however, the grade of concentrates was low to medium, averaging less than 60 per cent  $\text{MoS}_2$ . Test No. 3 is of special interest as it shows the results obtained by flotation of the same ore on water at different temperatures. The decided improvement in the grade of

---

<sup>a</sup> Wood, H. E., The Wood flotation process: Trans. Am. Inst. Min. Eng., vol. 44, 1912, p. 693.

concentrates by using warm water may be attributed to the decreased surface tension of the liquid allowing some particles of gangue to sink more readily. The recovery with warm water is, however, remarkable and can be accounted for only by some factor, such as a difference in the rate of feed, that would make the tests not strictly comparable. Detailed information as to the tests is given by Wood.<sup>a</sup> A summary of the data obtained by Wood is given in a table which is presented below:

*Results of concentration tests of molybdenite ores by Wood flotation process.*

Test No.	Size of feed.	Per cent MoS <sub>2</sub> .		Recovery, per cent.	Remarks.
		In feed.	In concentrates.		
1	Through 40-mesh..	2.08	41.38	99.00	Colorado ore. Retreatment would materially increase the grade of concentrates with little loss.
2	Through 20-mesh..	2.00	61.55	92.27	Alaskan ore.
3	.....	1.87	45.31 (a) 56.70 (b)	79.08 (a) 89.17 (b)	Canadian ore with a quartz and mica gangue. (a) was obtained with water at 56° F. and (b) with water at 110° F.
4	Through 20-mesh..	6.66	63.50	96.34	Foreign ore.
	Through 30-mesh..	6.73	67.42	86.26	
	Through 40-mesh..	6.66	65.50	86.48	
	Through 40-mesh..	8.95	74.60	94.19	
5	Through 40-mesh..	8.65	30.00 (a) 81.45 (b)	79.00 (a) 26.30 (b)	Alaskan ore containing pyrrhotite and magnetite, and low in silica. (a) was obtained with raw ore, and (b) with slightly roasted ore.

In order to obtain data as to concentration of a molybdenite ore containing copper sulphides by this process a small-scale test was made of a sample of ore from the Whale claim, in Copper Canyon, near Copperville, Ariz. The ore consisted of a clean white quartz containing about 7 per cent of molybdenite and 2 per cent of copper, present largely as chalcopyrite.<sup>b</sup>

After the ore had been dried and then crushed in a ball mill to pass a 40-mesh screen, it was concentrated on a laboratory-size flotation machine of the Wood type. The concentrator was so arranged that the floating film of concentrates picked up by the take-off belt was discharged to a second flotation surface from which it was allowed to run into a concentrate tank. The material that sank in the second flotation tank was considered middlings. The ore and water feeds to the machine were adjusted with the idea of causing as much chalcopyrite as possible to sink. The results of the test are given in the accompanying table.

<sup>a</sup> Wood, H. E., The Wood flotation process: Trans. Am. Inst. Min. Eng., vol. 44, 1912, pp. 684-701.

<sup>b</sup> For a detailed description of this ore see p. 52.

*Results of water-flotation test with concentrator of the Wood type of molybdenite-chalcopyrite ore from the Whale claim, near Copperville, Ariz.*

Item.	Ore.	Concentrates.	Middlings.	Tailings.
Weight, grams.....	4,000.00	538.00	107.00	<sup>a</sup> 3,355.00
Per cent MoS <sub>2</sub> .....	<sup>b</sup> 7.10+	47.44	6.74	.65
Per cent Cu.....	<sup>b</sup> 1.93+	5.26	5.90	1.27
Weight MoS <sub>2</sub> , grams.....	284.25	255.23	7.21	21.81
Weight Cu, grams.....	77.22	28.30	6.31	42.61
Extraction MoS <sub>2</sub> , per cent.....		89.79	2.54	.....
Extraction Cu, per cent.....		36.65	8.17	.....
Loss MoS <sub>2</sub> , per cent.....				7.67
Loss Cu, per cent.....				55.18

<sup>a</sup> 467 grams of material lost in the overflow was assumed to assay the same as the 2,888 grams of tailings recovered.

<sup>b</sup> Calculated.

The great difference between the percentage of recovery of molybdenite and of chalcopyrite is of particular interest. As compared with the feed, the concentrates contained approximately seven times the percentage of MoS<sub>2</sub> and less than three times the percentage of copper, and the recovery in the concentrates was 89.79 per cent of the molybdenite, as compared with only 36.65 per cent of the copper. Inasmuch as the crushed ore had stood only a short time before treatment, so that the particles of chalcopyrite had little opportunity to oxidize, which would cause them to be more readily wetted, the author considers the widely different results obtained in the concentration of the two minerals to be remarkable.

It is of interest to note that a water-flotation process for the concentration of molybdenite ore is being used at present on a commercial scale by Henry E. Wood & Co., of Denver, Colo.

Methods of treating both the original ore and the concentrates to remove pyrite, chalcopyrite, pyrrhotite, and other metallic sulphides and such ingredients as mica that are likely to float with the molybdenite are discussed later.

#### OIL-FLOTATION PROCESSES.

Many of the principal processes of concentration by oil flotation are described in detail by Hoover<sup>a</sup> and in numerous articles that have recently appeared in the mining press. Published data giving the results obtained by the application of these processes to molybdenite ores are extremely meager, and are practically confined to the results of treatment by the Elmore vacuum flotation system. As the Bureau of Mines at present has no equipment at its disposal for conducting tests of the various flotation processes, the concentration of molybdenite by these methods must, of necessity, be discussed here in only a general way.

<sup>a</sup> Hoover, T. J., Concentrating ores by flotation, 1912, 221 pp.



Broadly speaking, oil flotation probably offers the best method of treating molybdenite ores in general, and those particular processes that have been successfully applied in concentrating ores of the other metallic sulphides would, without doubt, meet with equal success in the treatment of molybdenite. The above statement does not mean that all molybdenite ores can be successfully concentrated by oil flotation, for the physical characteristics and mineralogical composition of an ore are as important factors in determining the success of any oil-flotation process as of electrostatic or water-flotation methods. The ores best suited for treatment by oil flotation are those in which the molybdenite is flaky. From ores in which the mineral is so fine as to be almost amorphous and the gangue contains a considerable proportion of soft material, such as kaolinized feldspar, it is almost impossible to obtain a good grade of concentrates because of the flotation of some of the finer particles of gangue. Of course, if molybdenite is associated with other metallic sulphides, such as chalcopyrite, pyrite, or pyrrhotite, these are concentrated too, and must either be removed from the ore, by a preliminary treatment, or from the concentrate if a high-grade product is desired. Methods that may be employed for this purpose are discussed later. Ores containing magnetite, hematite, garnet, hornblende, or similar gangue minerals that are good conductors of electricity; and therefore are not adapted for concentration by electrostatic methods, are especially suited for treatment by flotation.

Laboratory experiments indicate that in general the particles of molybdenite that can be floated are much coarser than those of other metallic sulphides, and if finer crushing is not necessary for the liberation of the molybdenite from the gangue, material as coarse as 20 mesh may be successfully concentrated. On the other hand, there is every reason to believe that a good recovery and a fair grade of concentrates can be obtained from the treatment of molybdenite through 200 mesh because galena and sphalerite of similar fineness have been successfully treated by oil flotation. In general, therefore, oil-flotation processes are applicable to a wider range of sizes than are electrostatic or water-flotation methods, and many of them have the added advantage of large capacities from small units. For example, Hoover estimates that in one type of apparatus used in the minerals-separation process a unit consisting of six mixing boxes each only 16 inches wide and 36 inches deep will have a daily capacity of 50 to 60 tons of ore,<sup>a</sup> presumably Broken Hill lead-zinc tailings, and a 5-foot Elmore machine will ordinarily treat from 25 to 45 tons of crude ore in 24 hours.<sup>b</sup> Capacities equal to and perhaps even greater than these could probably be obtained in the treatment of molybdenite ores by the same units, but no definite data either as to capaci-

<sup>a</sup> Hoover, T. J., *Concentrating ores by flotation*, 1912, p. 120.

<sup>b</sup> Hoover, T. J., *op. cit.*, p. 101.



ties or as to costs of any oil-flotation process when applied to molybdenite ore have, to the author's knowledge, been made public. In fact, owing to the policy of secrecy adopted by most companies interested in the development of oil-flotation methods, little information is available regarding the economics of the various processes, even as more commonly applied to the concentration of the sulphides of copper, lead, and zinc. However, comprehensive data with regard to the costs of treating Broken Hill lead-zinc tailings by four well-known processes, namely, the Potter-Delprat, De Bavay, Elmore, and minerals separation, are given by Hoover.<sup>a</sup> His estimates of 3s. 6d. (\$0.85) per ton as the average cost of flotation and 9s. 3d. (\$2.25) per ton as the average total cost of re-treating these tailings on a large scale form a poor basis on which to estimate the cost of treating molybdenite ores by the same processes in small plants such as might be installed at mines producing 50 or 100 tons of ore a day. It is safe to say, however, that the costs of small-scale concentration of molybdenite ores would be considerably higher than those given above.

## RESULTS WITH ELMORE VACUUM FLOTATION.

The following table shows the results obtained in concentration tests of two molybdenite ores by the Elmore vacuum-flotation system. It will be noted that in the first test, in which the gangue was largely feldspar, the percentage recovery was high and the concentrate obtained was only medium grade, probably because of the flotation of some of the finer particles of light feldspar. In the second test, in which the gangue was relatively heavy, chiefly magnetite and garnet, both the grade of concentrates and the recovery obtained were materially better.

*Concentration tests on molybdenite ore by the Elmore vacuum-flotation process.<sup>a</sup>*

Per cent MoS <sub>2</sub> .			Recovery, per cent.	Remarks.
In ore.	In tailings.	In con- centrates.		
5.67	0.42	68.00	93.2	Gangue principally feldspar.
3.83	.10	85.95	98.1	Gangue principally magnetite and garnet.

<sup>a</sup> Converted from figures given by A. S. Elmore, Vacuum flotation process for concentration: Eng. and Min. Jour., vol. 83, 1907, pp. 908, 909.

It is reported that the Elmore process is being used for treating molybdenite at two mines in Norway and that large quantities of concentrates have already been shipped.<sup>b</sup> The writer believes that these plants are as yet the only ones in which any type of oil concen-

<sup>a</sup> Hoover, T. J., op. cit., pp. 164-166.

<sup>b</sup> Molybdenum recovery by Elmore process: Eng. and Min. Jour., vol. 99, 1915, p. 907.

tration has been attempted for concentrating molybdenite on a commercial basis.

The results of several large-scale concentration tests by the Ore Concentration Co. (Ltd.), of London, of four different lots of molybdenite ore by the Elmore process are given in the accompanying table. The results of these tests, which were made with ore crushed through 30 mesh, compare favorably with those given in the preceding table and indicate in general what might be expected in the treatment of any molybdenite ore to which the process is adapted.

*Concentration tests of molybdenite ore by the Elmore vacuum-flotation process.<sup>a</sup>*

Weight of ore treated, tons.	Per cent MoS <sub>2</sub> in concentrates.	Recovery, per cent.
32.0	85.68	91.5
32.0	86.30	94.5
37.0	84.25	89.7
3.5	83.00	90.0
104.5	84.80	91.7

<sup>a</sup> Molybdenum recovery by Elmore process; Eng. and Min. Jour., vol. 99, 1915, p. 907.

#### REMOVAL OF ACCOMPANYING METALLIC SULPHIDES.

When other metallic sulphides, such as chalcopyrite, pyrite, or pyrrhotite, accompany molybdenite they are concentrated with the latter mineral by both electrostatic and flotation methods. The presence of pyrite or pyrrhotite in the concentrates is undesirable only in that it lowers the grade of the product, but chalcopyrite, bornite, or other copper sulphides are objectionable in that if the concentrates contain more than 1 or 2 per cent of copper they are practically unsalable, and even small percentages of copper greatly lower their value.

Various methods have been suggested for removing the other sulphides either before or after the concentration of the molybdenite. Most of these methods depend on the ore or concentrates being lightly roasted. In this way the larger part of the pyrite and chalcopyrite are rendered magnetic and may be removed by means of a magnetic separator, or, through the formation of a coating of oxide on the surface of each particle, the pyrite and chalcopyrite in the ore are readily wetted and sink if the material is subjected to flotation. It is at once evident that these new properties imparted to the pyrite and chalcopyrite by roasting can be taken advantage of either separately or in a combination of processes. In the writer's opinion it is cheaper and more effective as regards ultimate recovery of molybdenite to remove the other sulphides after concentration rather than before. No matter how carefully conditions of roasting are controlled some

of the molybdenite will be partly converted to oxide on the surface of the individual grains, and this coating will entail some loss of the molybdenite whether the material is treated by flotation or on an electrostatic separator. So far as recovery of the molybdenite is concerned it is, therefore, advisable to roast the combined concentrates and remove the oxidized pyrite and chalcopyrite by magnetic separation.

On the other hand, even with the most carefully regulated roasting, all the pyrite and chalcopyrite do not become magnetic. If the non-magnetic product from the magnetic separator contains an appreciable amount of copper its retreatment by flotation may be advisable in order to further rid the molybdenite from grains of chalcopyrite and pyrite which, although nonmagnetic, are covered with a film of oxide that will cause them to become wet and to sink. In other cases it may be advisable to dispense entirely with magnetic separation and to treat the lightly roasted concentrate directly by flotation.

A method for separating chalcopyrite, which takes advantage of the oxidation of the surface of the particles at ordinary temperatures, is suggested by a correspondent of the *Engineering and Mining Journal*. In regard to a molybdenite ore containing chalcopyrite, he states that: "After grinding the ore to 40 mesh, it was dampened and dried. Under these conditions 90 per cent of the  $\text{MoS}_2$  will float, and practically every particle of the chalcopyrite sinks. This furnishes a cheaper method than to roast the mixed flotation concentrates with the intention of removing the copper ore magnetically."<sup>a</sup>

Wood suggests treating the combined concentrates on the ordinary type of wet concentrating table with the idea of separating the molybdenite in what is usually the tailing product.<sup>b</sup> The author believes that this method would not be particularly effective. Moreover, inasmuch as the presence of even a small fraction of 1 per cent of copper is objectionable in the concentrates, it is doubtful whether any of the processes suggested above are commercially capable of reducing the copper to 0.1 per cent., for example, in any material in which the ratio of copper sulphide to molybdenite is relatively large. For such ore chemical methods of separation will probably have to be used.

#### SUMMARY.

Each particular molybdenite ore presents its own concentration problems. Usually one of the methods, or a combination of the methods, discussed above will give satisfactory results, but in some instances other ore-dressing processes must be employed in conjunction with them.

<sup>a</sup> Molybdenum, tungsten, and uranium ores, *Eng. and Min. Jour.*, vol. 97, 1913, p. 114.

<sup>b</sup> Wood, H. E., The Wood flotation process: *Trans. Am. Inst. Min. Eng.*, vol. 44, 1912, p. 693.



**CONCENTRATION OF WULFENITE ORES.**

Unlike the concentration of molybdenite, the treatment of wulfenite ores presents few difficulties. Not only are the ordinary wet processes of concentration by jigs, tables, vanners, slimers, etc., such as are usually employed in the treatment of ores of lead, copper, and zinc, adapted to the concentration of wulfenite, but, when properly applied, they result in a high extraction of the mineral. Moreover, even the finest particles of wulfenite are readily wetted; hence sliming of the ore does not occasion the considerable losses that would occur under similar circumstances in the treatment of most metallic sulphides, such as galena and chalcopyrite, as these minerals float readily when finely divided. On account of the high specific gravity of wulfenite (6.7 to 7) it is readily separated from all of the gangue minerals with which it occurs. It is, however, frequently associated with vanadinite (specific gravity, 6.66 to 7.23), from which it can not be separated by wet methods, also with cerussite (specific gravity, 6.46 to 6.57), and anglesite (specific gravity, 6.12 to 6.39), from which it can be only partly separated. Occasionally small quantities of other lead minerals of high specific gravity, such as galena (specific gravity, 7.4 to 7.6), pyromorphite (specific gravity, 6.5 to 7.1), and mimetite (specific gravity, 7.0 to 7.25), occur with wulfenite and are then recovered in the wulfenite concentrate. The presence or absence of these other heavy minerals in the ore determines largely the grade of wulfenite concentrates that can be made. Theoretically, a concentrate consisting entirely of pure wulfenite contains 39.23 per cent molybdic trioxide ( $\text{MoO}_3$ ), and with many ores a product with 34 to 36 per cent molybdic trioxide is easily obtained, whereas with others a concentrate containing even 20 per cent molybdic trioxide is obtained with difficulty owing to the presence of considerable percentages of vanadinite, cerussite, etc. The costs of concentrating wulfenite ores by wet processes are in general comparable to those of treating ores of galena, sphalerite, chalcopyrite, etc., by similar methods.

Many deposits of wulfenite ore are situated at considerable distances from water supplies of sufficient magnitude to furnish the quantities requisite for wet concentration. It is, therefore, of interest to note that wulfenite can be concentrated successfully by pneumatic processes, and that there are pneumatic separators of various types, such as tables and jigs, now on the market that without doubt can effect nearly as high an extraction and make as good grade of concentrates as can be obtained by wet methods. The cost of separation by pneumatic processes is, of course, greater than by wet methods.



# CONCENTRATION TESTS OF WULFENITE ORE FROM OLD YUMA MINE, NEAR TUCSON, ARIZ.

The tables following give the results obtained by the author from a series of concentration tests of wulfenite ore from the Old Yuma mine, 14 miles northwest of Tucson, Ariz. The tests were not conducted with the idea of deriving the best method of treatment for the ore in question, but to show the wide range of processes that may be successfully applied to wulfenite ores in general. The ore used was of high grade, containing approximately 6 per cent molybdc trioxide. The gangue was an iron-stained silicious rock in which the wulfenite was present in crystals varying in size from minute specks up to one-half inch in maximum diameter. A sample of the original ore weighing 120 pounds was crushed in a small jaw crusher to pass through a 3-mesh screen, and divided into six lots by careful sizing. The first three of these lots, embracing material through 3-mesh on 20-mesh, were treated in a hydraulic pulsator jig. The next two lots, embracing material through 20-mesh on 100-mesh, were concentrated in a pneumatic jig, and the last lot of material through 100-mesh was run on a concentrating table of common make.

*Concentration tests, on machines of various types, of wulfenite ore from the Old Yuma mine, 14 miles northwest of Tucson, Ariz.*

TEST 1. CONCENTRATION ON HYDRAULIC PULSATOR JIG.

Item.	Through 3-mesh, on 6-mesh.				Through 6-mesh, on 10-mesh.			
	Ore.	Concen- trates.	Mid- dlings.	Tailings.	Ore.	Concen- trates.	Mid- dlings.	Tailings.
Weight, pounds.....	32.50	3.54	1.46	27.50	27.00	4.73	1.00	21.27
Per cent MoO <sub>3</sub> .....	<sup>a</sup> 4.59+	36.43	8.46	.29	<sup>a</sup> 6.14+	32.76	5.13	.37
Weight MoO <sub>3</sub> , pounds	1.494	1.290	.124	.080	1.680	1.550	.051	.079
Extraction MoO <sub>3</sub> , per cent.....		86.35	8.30			92.26	3.04	
Loss MoO <sub>3</sub> , per cent..				5.35				4.70

Item.	Through 10-mesh, on 20-mesh.				Total.			
	Ore.	Concen- trates.	Mid- dlings.	Tailings.	Ore.	Concen- trates.	Mid- dlings.	Tailings.
Weight, pounds.....	18.00	4.55	1.57	11.88	77.50	12.82	4.03	60.65
Per cent MoO <sub>3</sub> .....	<sup>a</sup> 10.01+	34.98	11.45	.25	<sup>a</sup> 6.42+	<sup>a</sup> 34.57+	<sup>a</sup> 8.80+	<sup>a</sup> .31+
Weight MoO <sub>3</sub> , pounds	1.802	1.592	.180	.030	4.975	4.432	.355	.189
Extraction MoO <sub>3</sub> , per cent.....		88.35	9.99			89.07	7.13	
Loss MoO <sub>3</sub> , per cent..				1.66				3.80

<sup>a</sup> Calculated.

Concentration tests, on machines of various types, of wulfenite ore from the Old Yuma mine 14 miles northwest of Tucson, Ariz.—Continued.

## TEST 2. CONCENTRATION ON PNEUMATIC PULSATOR JIG.

Item.	Through 20-mesh, on 60-mesh.			Through 60-mesh, on 100-mesh.			Total.		
	Ore.	Concentrates.	Tailings.	Ore.	Concentrates.	Tailings.	Ore.	Concentrates.	Tailings.
Weight, pounds.....	21.00	4.05	16.95	7.00	0.80	6.20	28.00	4.85	23.15
Per cent $\text{MoO}_3$ .....	<sup>a</sup> 7.60+	33.48	1.42	<sup>a</sup> 3.94+	29.23	.67	<sup>a</sup> 6.68+	<sup>a</sup> 32.78+	<sup>a</sup> 1.22+
Weight $\text{MoO}_3$ , pounds.....	1.597	1.356	.241	.276	.234	.042	1.873	1.590	.283
Extraction $\text{MoO}_3$ , per cent.....		84.91			84.78			84.89	
Loss $\text{MoO}_3$ , per cent.....			15.09			15.22			15.11

## TEST 3. CONCENTRATION ON DIAGONAL-RIFFLE OSCILLATING TABLE (WET).

Item.	Through 100-mesh.		
	Ore.	Concentrates.	Tailings.
Weight, pounds.....	11.50	0.78	10.72
Per cent $\text{MoO}_3$ .....	<sup>a</sup> 2.33+	21.98	.91
Weight $\text{MoO}_3$ , pounds.....	.269	.171	.098
Extraction $\text{MoO}_3$ , per cent.....		63.57	
Loss $\text{MoO}_3$ , per cent.....			36.43

<sup>a</sup> Calculated.

## SUMMARY.

Item.	Ore.	Concentrates.	Middlings.	Tailings.
Total weight, pounds.....	117.00	18.45	4.03	94.52
Total weight $\text{MoO}_3$ , pounds.....	7.118	6.193	.355	.570
Per cent $\text{MoO}_3$ .....	<sup>a</sup> 6.08+	<sup>a</sup> 33.56+	<sup>a</sup> 8.80+	<sup>a</sup> .60+
Extraction $\text{MoO}_3$ , per cent.....		87.00	4.99	
Loss $\text{MoO}_3$ , per cent.....				8.01

<sup>a</sup> Calculated.

Distribution of wulfenite, and recoveries and losses, according to sizes in tests 1, 2, and 3.

Machine used.	Size of material.	Per cent of total $\text{MoO}_3$ in—			
		Feed.	Concentrates.	Middlings.	Tailings.
Hydraulic pulsator jig...	Through 3-mesh, on 6-mesh.....	20.98	18.12	1.74	1.12
	Through 6-mesh, on 10-mesh.....	23.60	21.77	.72	1.11
	Through 10-mesh, on 20-mesh.....	25.32	22.37	2.53	.42
	Through 3-mesh, on 20-mesh.....	69.90	62.26	4.99	2.65
Pneumatic pulsator jig...	Through 20-mesh, on 60-mesh.....	22.44	19.05		3.39
	Through 60-mesh, on 100-mesh.....	3.88	3.29		.59
	Through 20-mesh, on 100-mesh.....	26.32	22.34		3.98
	Through 100-mesh.....	3.78	2.40		1.38
Concentrating table.....					
Total.....		100.00	87.00	4.99	8.01

## DISCUSSION OF RESULTS OF TESTS.

The results obtained by treatment on the hydraulic pulsator jig show that the concentrates obtained averaged 34.57 per cent molybdc trioxide ( $\text{MoO}_3$ ), which corresponds to 88.12 per cent pure wulfenite, and that the extraction in these concentrates was 89.07 per cent. The total middlings, which amounted to only 4.03 pounds, or 5.2 per cent of the feed, accounted for 7.13 per cent of the remaining molybdenum, and the bulk of the wulfenite contained in them could without doubt have been recovered as a high-grade concentrate by recrushing and treating the product on concentrating tables, even though the middlings contained considerable cerussite. It will be noted that the tailings averaged only 0.31 per cent molybdc trioxide, and although they amounted to 78.26 per cent by weight of the feed, the loss of wulfenite in them was only 3.8 per cent of the total quantity in the feed. In this instance, the ore through 10-mesh and on 20-mesh was jigged only to show that the method was applicable to treating material of this fineness. In commercial work it would probably be preferable to treat this size of wulfenite ore on concentrating tables, as in general more expert attention is required in the successful operation of jigs than in the use of tables.

The tests on the air pulsator jig were made to indicate the applicability of pneumatic processes to wulfenite ores in deposits situated at such distances from water supplies as to make wet concentration out of the question. A summary of the tests shows that the concentrates averaged 32.78 per cent molybdc trioxide, corresponding to 83.56 per cent pure wulfenite, and that the extraction in the concentrates was 84.89 per cent. No middling products were made, and for that reason the tailings were richer in wulfenite than they otherwise would have been. At the end of the two runs a perfect separation of the concentrates and tailings remaining in the jig was impossible, and in the effort to keep the concentrates clean a small quantity of them was skimmed off with the tailings. This procedure also lowered the extraction. Further, closer sizing of the material before treatment would, without doubt, have been beneficial. However, the tests demonstrated that an excellent grade of concentrates can be made by a pneumatic process, and considering all the conditions, the average extraction of 84.89 per cent compares favorably with the average extraction of 89.07 per cent made from the coarser sizes of material by the hydraulic pulsator jig, and it is probably little less than that which would have been obtained if the material had been treated by wet concentration on tables.

A common make of oscillating table with riffles terminating in a diagonal line across the deck was used for the wet concentration of the material that passed through a 100-mesh screen. A good

marketable grade of concentrates was made, but the extraction (63.57 per cent) was poor. The results indicate that a considerable porportion of the material was too fine to be treated efficiently on this type of machine and that the ore should be further sized or classified and the finer part treated on a slime table.

Summarizing the results of the three series of tests, it will be seen that the average molybdic trioxide content of the total concentrates, total middlings, and total tailings was 33.56 per cent, 8.80 per cent, and 0.60 per cent, and that the extraction in the concentrate was 87 per cent, and in the middlings 4.99 per cent, whereas the total loss in the tailings was 8.01 per cent.

The table giving the distribution of wulfenite and the recoveries and losses in the various sizes of material, shows that practically 70 per cent of the total wulfenite in the ore was contained in the material through 3-mesh and on 20-mesh, and that out of this amount 62.26 per cent was recovered in the concentrates from the hydraulic pulsator jig, whereas 4.99 per cent more was contained in the middlings from the jig, a total recovery of 67.25 per cent. The table further shows that out of 26.32 per cent of the total wulfenite in the ore contained in the material through 20-mesh and on 100-mesh, the pneumatic pulsator jig recovered 22.34 per cent. Only 3.78 per cent of the wulfenite content of the ore was finer than 100-mesh, and of this amount the concentrating tables recovered 2.4 per cent.

Although the tests discussed above were not conducted with the idea of deriving a method of treatment, the results indicate in a general way the procedure that should be adopted. In the author's opinion the ore could be treated efficiently by wet concentration approximately as follows:

After the ore has been crushed in a jaw or crusher in gyratory crushers to about  $\frac{1}{2}$ -inch size it should be further crushed in rolls to about  $\frac{1}{4}$ -inch and the product screened into three series of sizes. Approximately, these sizes should be as follows: Through 4-mesh and on 8-mesh, and through 8-mesh and on 12-mesh, these two sizes to be treated on jigs; through 12-mesh and on 20-mesh, through 20-mesh and on 40-mesh, through 40-mesh and on 80-mesh, and through 80-mesh and on 120-mesh, these four sizes to be treated on concentrating tables, and the material through 120-mesh to be treated on slime tables. The jigs should be operated so as to obtain as clean concentrates and tailings as possible, and the middlings, after recrushing (perhaps through 20-mesh), should be appropriately sized for reconcentration by the tables.

In the operation of the tables the price received for various grades of wulfenite concentrates would determine where the cut between concentrates and tailings should be made. It might be advanta-





A. TAILING PILES CONTAINING WULFENITE, FROM STAMP AND CYANIDE MILLS, MAMMOTH, ARIZ. LOADING TUNNEL AND TRACK TO WULFENITE MILL IN THE FOREGROUND.



B. BOYKIN & HEREFORD WULFENITE MILL, MAMMOTH, ARIZ.

LIBRARY  
OF THE  
UNIVERSITY OF ILLINOIS

geous, from a commercial standpoint, to sacrifice something on the recovery in order to make an especially high grade of concentrates.

The author is confident that if the process described above was applied on ore similar to that treated it would result in at least a 90 per cent recovery of the wulfenite as a high-grade concentrate containing 30 per cent or more of molybdc trioxide.

It is of interest to note that the particular lot of ore used in the tests assayed 3.64 ounces of gold per ton and that a microscopic examination showed that much of the gold was contained as native metal in the wulfenite crystals themselves. As far as known, this is the first time that the presence of visible gold has been reported in wulfenite. Further, it was ascertained that the gold was seemingly confined to dark-colored crystals and that the more plentiful orange-colored crystals were free from the element. In order to verify the results of the microscopic examination, two lots of crystals were hand picked from the coarser sizes of concentrates obtained by jigging. One lot consisted of clear, orange-yellow crystals, and the other of dark-brown to almost black crystals. These lots were crushed and assayed. Two assays of the yellow material failed to show even a trace of gold, whereas two assays of the dark crystals yielded 143.32 and 138.02 ounces per ton, thus proving conclusively that the gold was confined to the latter crystals. The author believes that a considerable proportion of the dark color of these crystals is due to minute particles of gold.

It is regretted that owing to the removal of considerable material from the coarser sizes of jigged concentrates for experimental purposes, no statement can be made regarding the recovery of gold in the concentrates as a whole. However, it would probably be somewhat less than the corresponding recovery of wulfenite or of molybdc trioxide, as no doubt the cerussite in the ore contains some gold. For example, the concentrates and tailings derived by jigging the material through 20-mesh on 60-mesh assayed 15.36 and 0.82 ounces of gold per ton. Calculated from these assays, the gold content of the feed was 3.62 ounces per ton, and the recovery in the concentrates was 81.74 per cent, or slightly less than the corresponding recovery of molybdc trioxide, which was 84.91 per cent. The percentage of gold recovered, however, is probably almost identical with that of lead.

#### **BOYKIN & HEREFORD WULFENITE MILL AT MAMMOTH, ARIZ.**

At Mammoth, Pinal County, Ariz., there are large piles of tailings (Pl. XVIII, A) derived from ore from the Mammoth mine, about 3 miles distant at Schultz.<sup>a</sup> This ore was originally crushed in a stamp

---

<sup>a</sup> For description of the ore from this mine see p. 47.

mill and treated by amalgamation to recover its free-gold content. Later the tailings from this process were cyanided to extract the considerable part of the gold that was not free milling or had otherwise been lost in the first operation. Subsequently part of the resulting tailings was sluiced to recover the wulfenite content. This last operation was over a decade ago, and it probably supplied about 750 tons of wulfenite concentrates out of the 795 tons of combined wulfenite and molybdenite concentrates that constituted the total production of this county in 1903.<sup>a</sup>

A part of these tailings piles is shown in Plate XV, *B*. They are estimated to contain 200,000 to 250,000 tons of material and 1 to 2 per cent of wulfenite. In 1914 Messrs. F. H. Hereford and R. O. Boykin, of Tucson, Ariz., who had obtained control of all but about 25,000 tons of this material, installed two concentrating tables in the mill building of the old cyanide plant and commenced the re-treatment of the tailings to recover the wulfenite. Operations were so successful that early in 1915 they entirely remodeled the small plant and increased the number of concentrating tables to seven. At this time (June, 1915) operations are being conducted as follows:

The tailings are dug from the tailings piles by means of a four-horse scraper and dragged up an incline to a loading platform, where they are dumped onto a coarse grizzly, which removes pieces of board, brush, etc., and allows the tailings to fall through into a car. The scraper used and the loading platform are shown in Plates XV, *B*, and XVIII, *A*. The cars into which the tailings are dumped, each having a capacity of slightly over a ton, are trammed by hand for several hundred feet to the foot of an incline leading to the mill bin. Here each car is hooked onto a cable and drawn up the incline to the upper edge of the bin, about 30 feet above the ground, and its contents dumped into the bin by hand. Each empty car is returned down the incline by gravity and run onto a switch to allow the next loaded car to pass and then trammed back into the tunnel beneath the loading platform to be refilled. Plate XVIII, *B*, shows a general view of the mill and its surroundings. It is understood that the work of scraping the tailings and placing them in the mill bin is let by contract and that the cost is about 14 cents per ton.

#### DISCUSSION OF FLOW SHEET.

In the following description of the operation of the mill the numbers refer to the flow sheet shown in figure 2. The mill bin or feed hopper, *1*, has the form of an inverted square pyramid truncated parallel with its base, and holds about 40 tons. It is clearly shown in Plate XVIII, *B*. From this hopper the tailings are washed by a

<sup>a</sup> Pratt, J. H., The steel-hardening metals: Mineral Resources U. S. for 1903, U. S. Geol. Survey, 1904, p. 308.



small stream of water over an inclined screen, 2, of about 4-mesh. The oversize on this screen is removed by hand and thrown away, and the undersize is carried through a launder into the mill building, where it discharges into a hindered-settling classifier, 3. The overflow from this classifier goes to a second hindered-settling classifier, 4, and the overflow from the second classifier to a third, 5, and so on through five classifiers. The overflow from the fifth classifier passes into a settling cone, 8, the overflow from which goes into the tailings

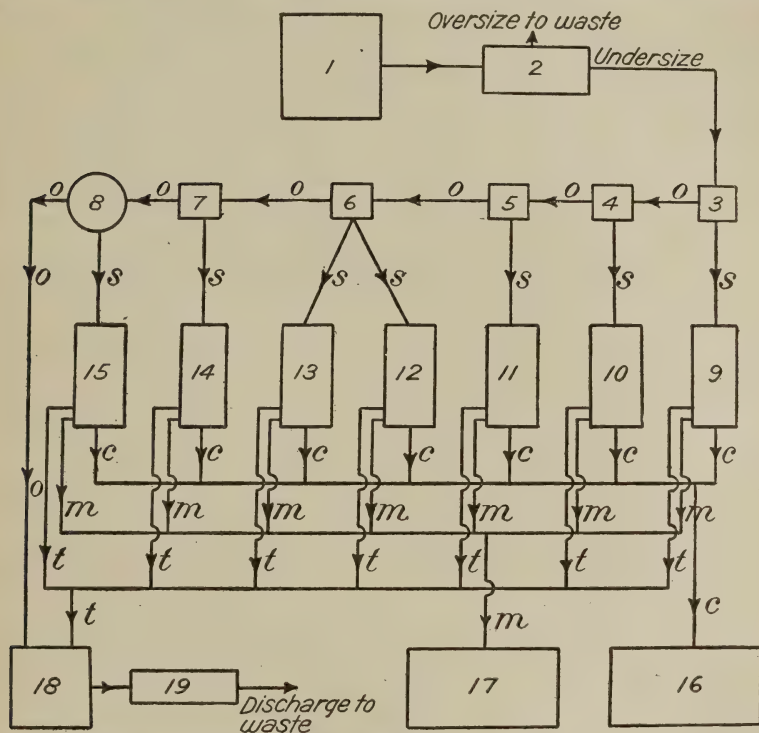


FIGURE 2.—Flow sheet of Boykin & Hereford wulfenite mill, Mammoth, Ariz. 1, Mill bin or hopper, having a capacity of about 40 tons; 2, inclined 4-mesh screen; 3 to 7, hindered-settling classifiers; 8, settling cone; 9 to 11, Overstrom tables; 12, Ludwig table; 13 to 15, Card tables; 16, concentrate storage; 17, middlings storage; 18, tailings sump; 19, Byron Jackson 3-inch "Dreadnaught" sand pump.

sump, 18. The spigot products from the first three classifiers, 3 to 5, are treated separately on three Overstrom tables, 9 to 11. The spigot product from the fourth classifier, 6, is divided into two parts, one of which goes to a Ludwig table, 12, and the other to a Card table, 13. The spigot products from the fifth classifier, 7, and from the settling cone, 8, are treated separately on two Card tables, 14 and 15. Each of the concentrating tables makes a middlings product, which goes to a storage bin, 17, for re-treatment. It is understood that these middlings are sometimes re-run on one of the concentrating tables, which is temporarily cut out of the mill system for their re-treatment, and

sometimes they are returned to the mill system as a whole. The tailings from all the tables go to the tailings sump 18, and the concentrates to storage bin 16, from which they are taken to be dried and sacked. The tailings are pumped from the tailings sump 18 by a Byron Jackson 3-inch "Dreadnaught" sand pump, 19, and discharged through a pipe onto a tailings pile a short distance from the mill.

Power is furnished by two Fairbanks-Morse oil engines of 12 and 20 horsepower. The smaller engine drives the sand pump 19, and the larger engine supplies power for operating the concentrating tables 9 to 15, a Byron Jackson 4-inch pump that supplies the plant with water, and a Browning 2-kilowatt dynamo which is rated to supply 16.6 amperes at 120 volts when run at 1,425 revolutions per minute. The dynamo furnishes electricity for lighting the mill and for illuminating the scraping and tramming operations on the tailings piles at night. The mill operates 24 hours a day and employs 10 to 12 men.

It is reported that about 2 tons of concentrates are obtained each day from the treatment of about 200 tons of tailings. A large sample of these concentrates, furnished to the author through the courtesy of Messrs. Hereford and Boykin, upon analysis yielded the results shown in the following table:

*Analysis of wulfenite concentrates from mill at Mammoth, Ariz. <sup>a</sup>*

Constituent.	Per cent.
MoO <sub>3</sub> .....	22.62
PbO.....	61.83
V <sub>2</sub> O <sub>5</sub> .....	1.28
Fe <sub>2</sub> O <sub>3</sub> and Al <sub>2</sub> O <sub>3</sub> .....	5.50
CuO.....	.65
SiO <sub>3</sub> .....	1.53
P <sub>2</sub> O <sub>5</sub> .....	.30
SO <sub>3</sub> .....	1.96
CO <sub>2</sub> .....	3.20
Cl.....	.23
CaO, K <sub>2</sub> O, etc.....	b .90
Total.....	100.00

The above analysis indicates the presence of the four principal mineral ingredients of the concentrates in the following percentages; Wulfenite, 57.63; cerussite, 18.32; vanadinite, 8.65; anglesite, 6.52; total, 91.12 per cent. As the last three minerals are of nearly the same specific gravity as wulfenite, the rather low molybdic trioxide content of the concentrates (22.62 per cent) is not surprising. In fact, with such relatively large percentages of these other heavy lead minerals in the feed, the writer does not see how any considerable betterment of the grade of concentrates could be expected. Inasmuch as over 1 per cent of tungsten had been reported to Messrs. Hereford

<sup>a</sup> Analyst, H. A. Doerner, Bureau of Mines.

<sup>b</sup> Approximate.

and Boykin in the analysis of a sample said to have represented a considerable tonnage of concentrates, it is of interest to note that none of that element was found in the analysis given above.

Although it would be difficult to improve greatly the grade of concentrates being made at this plant, the author is of the opinion that the capacity of a plant of this character could be greatly increased by the installation of an efficient screening system to remove the coarser particles of gangue before concentration on the tables, and that the extraction of wulfenite could be considerably improved by the treatment of the fines on slime tables or slime vanners.

#### RESULTS OF SCREEN TEST.

A screen test of a sample of concentrates was made, the results being presented below. The test showed that 93.6 per cent would pass through a 100-mesh screen, and that 76.2 per cent would pass through a 200-mesh screen. Further, it is reasonable to assume that the losses of wulfenite occurring in the mill are confined almost entirely to very fine material, so the relative proportion of fine to coarse wulfenite in the feed is probably even greater than indicated by the results of the screen test. It is obvious that, as the tables used for treating the finer sizes of material are not suited for treating slimes, a considerable loss of wulfenite must occur in their tailings. Moreover, the overflow from the settling cone in the mill is not treated at all but runs directly to the tailings sump. A panning test of this overflow product showed the presence of considerable fine wulfenite which could at least be partly recovered by allowing the overflow to settle in tanks and treating the slimes on slime tables or slime vanners. Further, as stated above, it is the author's opinion that a much better recovery would be effected if the finer part of the material, embracing the spigot product from the settling cone, and even, perhaps, the spigot product from the fifth, or final, classifier now being handled by the tables, were treated on concentrators adapted to handling slimes.

*Results of screen test on wulfenite concentrates from mill at Mammoth, Ariz.*

[Screen ratio, 1.414.]

Mesh.	Opening of holding screen.		Weight.	Per cent of total weight.	Per cent of cumulative weight.
	Inch.	Milli-meter.			
On 48.....	0.0116	0.295	<i>Grams.</i> 41	2.7	2.7
Through 48, on 65.....	.0082	.208	21	1.4	4.1
Through 65, on 100.....	.0058	.147	35	2.3	6.4
Through 100, on 150.....	.0041	.104	79	5.3	11.7
Through 150, on 200.....	.0029	.074	182	12.1	23.8
Through 200.....			1, 142	76.2	100.0
Total.....			1,500	100.0	.....

The results of the screen test showed that only 2.7 per cent of the concentrates were caught on a 48-mesh screen. In other words, the original feed to the mill might be screened through a 48-mesh sieve, and by throwing away the oversize only 2.7 per cent of the material now recovered would be lost. It is regretted that the results of a screen test on the present feed to the mill are not available, so that the exact proportion of the gangue that might be eliminated by screening with a practically negligible loss might be accurately stated. The writer, however, is confident that more than one-quarter, and perhaps even one-half, of the total feed to the mill could be removed by screening to 40-mesh. That the capacity of the mill would be considerably increased by such a procedure is obvious, and the recovery and the grade of concentrates would probably also be benefited. In fairness to the operators of the mill it must be said that the screening system at the time of the author's visit (screening to about 4-mesh at the outlet of the mill bin and to about 8-mesh or 10-mesh through pieces of screen placed in the feed boxes of the concentrating tables), was only a temporary makeshift, and has doubtless been replaced by a more efficient system.



## SELECTED BIBLIOGRAPHY ON MOLYBDENUM.

- ACKERMAN, EUGENE. Examen du marché de divers métaux—molybdène. Métaux et alliages, vol. 5, 1912, pp. 108–110. Describes uses of molybdenum, and lists principal localities in various countries from which its ores are derived.
- ANDREWS, E. C. Molybdenum. New South Wales Dept. of Mines and Agriculture, Geol. Survey, Mineral Resources No. 11, 1906, 17 pp. Describes occurrences of molybdenite and wulfenite ores in New South Wales.
- ANONYMOUS. Norwegian molybdenite. Eng. and Min. Jour., vol. 98, 1914, p. 820. Describes several Norwegian deposits and gives data as to production.
- Occurrence and uses of molybdenum ores. Bull. Imperial Inst., vol. 6, 1908, pp. 181–191. Summary of facts in regard to uses and to production and occurrence in various countries.
- Molybdenum, tungsten, and uranium ores. Eng. and Min. Jour., vol. 97, 1914, p. 114. Describes method of separating copper sulphides from molybdenite by flotation after superficial oxidation, obtained by wetting and drying.
- AUCHY, GEORGE. The rapid détermination of molybdenum in steel. Iron Age, vol. 70, Nov. 20, 1902, p. 4.
- BAAR, N. Über die Legierungen des Molybdäns mit Nickel des Mangans mit Thallium und des Kalzium mit Magnesium, Thallium, Blei, Kupfer, und Silber. Ztschr. anorg. Chem., vol. 70, 1911, pp. 352–394. Study of constitution of the alloys.
- BALL, L. C. Rare-metal mining in Queensland—résumé of recent field studies. Queensland Govt. Min. Jour., vol. 14, 1913, pp. 4–7. Describes principal molybdenite deposits.
- The wolfram, molybdenite, and bismuth mines of Bamford. Queensland Mines Dept., Geol. Surv. Pub. 248.
- BORNEMANN, K. Die Binären Metallegierungen—Legierungen des Molybdäns und des Wolframs. Metallurgie, vol. 9, 1912, pp. 384–392. Describes iron-molybdenum, nickel-molybdenum, iron-tungsten, alloys, etc.
- BROWN, A. P. The crystallization of molybdenite. Proc. Acad. Nat. Sci., Philadelphia, 1896, pt. 1, pp. 210–211.
- CAMERON, W. E. Wolfram and molybdenite mining. Queensland Govt. Min. Jour., vol. 5, 1904, pp. 62–65. History, statistics, and geology of principal Queensland deposits.
- Wolfram, molybdenite, and bismuth mining at Wolfram Camp, Hodgkinson goldfield. Queensland Govt. Min. Jour., vol. 4, 1903, pp. 350–352. Deals with geology, working, etc., of principal mines in this district of Queensland.
- CLENNELL, J. E. Molybdenum in cyanide solutions. Eng. and Min. Jour., vol. 97, 1914, pp. 363–364. Describes action of molybdenum in cyanide solutions.
- COOLIDGE, W. D. Some applications of wrought tungsten and molybdenum. Jour. Ind. and Eng. Chem., vol. 4, 1912, pp. 2–4. Describes uses of molybdenum for electrical contacts and winding electrical-resistance furnaces, etc.
- CROOK, A. R. Molybdenite at Crown Point, Washington. Bull. Geol. Soc. Am., vol. 15, 1904, pp. 283–288. Describes deposit and discusses petrographic relationships and accompanying minerals in molybdenum deposits in general.
- DARROCH, JAMES, and MEIKLEJOHN, C. A. A rapid method of determining molybdenum. Eng. and Min. Jour., vol. 82, 1906, p. 818. See Erratum, vol. 83, 1907, p. 177.
- DAVIS, R. S. Notes on technical analysis—analysis of wulfenite ores. Met. and Chem. Eng., vol. 9, 1911, pp. 458–459.

- DE BENNEVILLE, J. S. Some alloys of iron with molybdenum, tungsten, and chromium as solutions. *Jour. Am. Chem. Soc.*, vol. 16, 1894, pp. 735-757.
- Ternary alloys of iron with chromium, molybdenum, and tungsten. *Jour. Iron and Steel Inst.*, 1895, No. 1, pp. 202-248. A chemical study of the constitution of various alloys.
- DEBRAY, H. Recherches sur le molybdène. *Compt. rend.*, vol. 46, 1858, pp. 1098-1102. Describes early experiments in the production of metallic molybdenum.
- DITTS, E. J., and BOWMAN, R. G. The direct production of molybdenum steel in the electric furnace. *Trans. Am. Electrochem. Soc.*, vol. 20, 1911, pp. 355-372. Describes in detail experiments in direct electric smelting of molybdenite concentrates.
- EMMONS, W. H. Some ore deposits in Maine and the Milan mine, New Hampshire. *U. S. Geol. Survey Bull.* 432, 1910, pp. 21, 42, 47-49. Describes occurrence of molybdenite in several deposits.
- ESCARD, J. Méthodes électrochimiques et alumino-thermiques pour la préparation de la fonte de molybdène et des ferro-molybdènes. *La Lumière Électrique*, vol. 7, 1909, pp. 200-206. Describes preparation of pure molybdenum.
- EVANS, J. C. Determination of molybdenum in wulfenite. *West. Chemist and Metallurgist*, vol. 3, 1907, pp. 218-219.
- FAHRENWALD, F. A. A development of practical substitutes for platinum and its alloys, with special reference to alloys of tungsten and molybdenum. *Bull.* 109, *Am. Inst. Min. Eng.*, January, 1916, pp. 103-149. Describes investigation undertaken by research foundation of the National Dental Association. Pure ductile molybdenum coated with a precious metal or alloy in many ways superior to platinum or its alloys for dental work.
- FINK, C. G. Ductile tungsten and molybdenum. *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, pp. 229-234. Compares tensile strength, specific gravity, resistivity, etc., of tungsten and molybdenum wire.
- FRIEND, N. J., and MARSHALL, C. W. The influence of molybdenum on the corrodibility of steel. *Jour. Iron and Steel Inst.*, vol. 89, No. 1, 1914, pp. 503-507. Describes various tests in fresh, salt, and acid waters, etc.
- GIN, G. Memoir on the methods of treatment of simple and complex ores of molybdenum, tungsten, uranium, vanadium. Part I. Molybdenum. *Trans. Am. Electrochem. Soc.*, vol. 12, 1907, pp. 411-474. Deals chiefly with the manufacture of ferromolybdenum in the electric furnace.
- Sur le traitement de la wulfénite pour ferromolybdène a basse teneur en carbone. *Proc. 7th Int. Cong. Appl. Chem.*, vol. 10, 1909, p. 10. Describes method of treating wulfenite to obtain ferromolybdenum of low-carbon content.
- GLEDHILL, J. M. The development and use of high-speed tool steel. *Jour. Iron and Steel Inst.*, 1904, No. 2, pp. 127-182. General description covering heat treatment, tests, etc., of tool steels, including molybdenum-chromium and tungsten-molybdenum-chromium steels.
- GUICHARD, M. Sur la molybdenite et la préparation au molybdène. *Compt. rend.*, vol. 122, 1896, pp. 1270-1272. Describes the decomposition of molybdenite in the electric arc.
- GUILD, F. N. The composition of molybdite from Arizona. *Am. Jour. Sci.*, ser. 4, vol. 23, 1907, pp. 455-456. Gives analysis of mineral from Santa Rita mountains.
- GUILLET, LEON. Propriétés et constitution des aciers au molybdène, *Compt. rend.*, vol. 139, 1904, pp. 540-542. Describes experiments on various molybdenum steels, showing strength and hardening, and tempering properties.
- Recherches sur les aciers au molybdène. *Génie Civil*, vol. 45, 1904, pp. 242-244. Results of a series of systematic tests, with microphotographs of molybdenum steels.

- HÄNIG, A. Die modernen Eisenlegierungen und ihre Verwendung in der Eisen- und Stahlindustrie. Discusses ferromolybdenum. *Electrochem. Ztschr.*, Jahrg. 20, November, 1913, pp. 211-213. Discusses modern ferroalloys and their application in the iron and steel industry.
- HAYNES, ELWOOD. Alloys of cobalt with chromium and other metals. *Trans. Am. Inst. Min. Eng.*, vol. 44, 1912, pp. 573-577. Describes a cobalt-chromium-molybdenum alloy, and also a chromium-cobalt-tungsten-molybdenum alloy.
- HESS, F. L. Some molybdenum deposits of Maine, Utah, and California. *U. S. Geol. Survey Bull.* 340, 1908, pp. 231-240. Describes molybdenite deposits at Cooper, Catherines Hill, and other localities in Maine, and at Corona, Cal., also a wulfenite deposit at Alta, Utah.
- Molybdenum. *Mineral Resources U. S. for 1906*, U. S. Geol. Survey, 1907, pp. 527-529; *Mineral Resources U. S. for 1908*, U. S. Geol. Survey, 1909, pp. 745-747.
- HIBBARD, H. D. Manufacture and uses of alloy steels, *Bureau of Mines Bull.* 100, 1915, pp. 58-59. The use of molybdenum in high-speed tool steel is discussed.
- HILLEBRAND, W. F. Distribution and quantitative occurrences of vanadium and molybdenum in rocks of the United States. *Am. Jour. Sci.*, ser. 4, vol. 6, 1898, pp. 209-216; *U. S. Geol. Survey Bull.* 167, 1900, pp. 49-55.
- HILLS, B. W. The molybdenite deposits of Tunk Pond, Maine. *Min. World*, vol. 31, 1909, pp. 323-324. Describes deposits with special reference to their geology, lists accompanying minerals, and describes a phosphate of molybdenum called "knightite."
- INGERSOLL, C. A. On hemimorphic wulfenite crystals from New Mexico. *Am. Jour. Sci.*, ser. 3, vol. 48, 1894, pp. 193-195. Describes hemimorphic crystals from Turquoise mines in the Jarilla Mountains, Dona Ana County.
- JOHNSTON, R. A. A., and WILLIMOTT, C. W. Molybdenum and tungsten. *Mineral Resources of Canada*, *Geol. Survey of Canada, Bull.* 872, 1904, 16 pp. Describes various Canadian occurrences.
- JORISSEN, A. Sur la diffusion du molybdène dans le terrain houiller de Liège. *Bull. Soc. Chem. de Belgique*, vol. 27, 1913, pp. 21-25.
- KEENEY, R. M. The production of steels and ferroalloys directly from ore in the electric furnace. *Jour. Iron and Steel Inst.*, *Carnegie Scholarship Memoirs*, vol. 4, 1912, pp. 108-184. Describes production of molybdenum steel.
- Electric smelting of chromium, tungsten, molybdenum, and vanadium ores. *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, pp. 167-189. Gives historical and experimental data with regard to the manufacture of ferromolybdenum, pp. 184-188.
- LAUTSCH and TAMMAU, J. Über die Legierungen des Eisens mit Molybdän. *Ztschr. anorg. Chem.*, vol. 55, 1907, pp. 386-401. Discusses iron molybdenum alloys.
- LEHMER, C. Elektrisches verschmelzen sulfidischer Erze und Huttenprodukte unmittelbar auf Metall. *Metallurgie*, vol. 3, 1906, pp. 549-555 and pp. 596-602. Describes method of producing ferromolybdenum, chrome molybdenum, nickel molybdenum, manganese molybdenum, and copper molybdenum.
- LYON, D. A., KEENEY, R. M., and CULLEN, J. F. The electric furnace in metallurgical work. *Bureau of Mines Bull.* 77, 1914, pp. 146-151. Discusses electric smelting of molybdenite and the production of ferromolybdene.
- MENNICKE, HANS. Die Metallurgie des Wolframs, 1911, 416 pp. Treats of the preparation of molybdenum, ferromolybdenum, etc. (pp. 94-100).
- The metallurgy of molybdenum and vanadium. *Elektrochem. Ztschr.*, October, 1913.
- MEYERS, R. E. Commercial manipulation of refractory elements for incandescent-lamp purposes. *Trans. Am. Inst. Chem. Eng.*, vol. 3, 1910, pp. 172-187. Interesting data on ductile metallic molybdenum.



- MILLS, S. D. Molybdenite. Report of the Bureau of Mines, Ontario, 1902, pp. 45-48. Describes various deposits in Haliburton.
- MOISSAN, HENRI. Préparation et propriétés du molybdène pur fondu. *Compt. rend.*, vol. 120, 1895, pp. 1320-1326.
- The electric furnace. Translated from the French by Victor Lehuer, 1904, 305 pp. Deals with the preparation of metallic molybdenum (pp. 155-161).
- MOSES, A. J. The crystallization of molybdenite. *Am. Jour. Sci.*, ser. 4, vol. 17, 1904, pp. 359-364.
- NORTHROP, E. F. Tungsten and molybdenum—their thermal e. m. f. *Met. and Chem. Eng.*, vol. 11, January, 1913, p. 45. Gives temperature curve of the tungsten-molybdenum thermocouple from 1° to 1,000° C.
- OBLASKI, J. Report on the mines of the Province of Quebec for the year 1898. Dept. Colonization and Mines, 1899, pp. 15-17. Describes molybdenite deposits in Quebec.
- OHLY, J. Molybdenite; its determination, detection, and uses. *Min. Reporter*, vol. 50, 1904, pp. 10-11.
- PLUMMER, JOHN. Molybdenite deposits in Australia. *Min. World*, vol. 26, 1907, pp. 421-422. Describes deposits and methods of mining and preparation in New South Wales and Queensland.
- PIRANI, M. V., and MEYER, A. R. Über den Schmelzpunkt des Wolframs und des Molybdäns. *Ver. deut. phys. Gesell.*, vol. 14, 1912, pp. 426-428. The melting point of molybdenum is given as 2,450° C. + 30°, that of tungsten as 3,100° C. ± 60°.
- PORTEVIN, A. M. Contribution to the study of the special ternary steels. *Carnegie Scholarship Memoirs, Iron and Steel Inst. (London)*, vol. 1, 1909, pp. 275-276, 330-333. Gives tensile and shearing strength of various molybdenum steels and data on electrical resistance.
- PRATT, J. H. The steel and iron hardening metals, *Mineral Resources U. S. for 1904*, U. S. Geol. Survey, 1905, pp. 338-343.
- RUDER, W. E. Solubility of wrought tungsten and molybdenum. *Jour. Am. Chem. Soc.*, vol. 34, 1912, pp. 387-389. Gives data on solubility in common acids and alkalies.
- SAINT-SMITH, E. C. Molybdenite in the Stanthorpe-Ballandean districts, Southern Queensland. *Queensland Govt. Min. Jour.*, vol. 15, 1914, pp. 184-189.
- SCHALLER, W. T. Notes on powellite and molybdate. *Am. Jour. Sci.*, ser. 4, vol. 25, 1908, pp. 71-75. Describes powellite from Llano County, Tex., and Nye County, Nev., also molybdate from Hortense, Colo.
- The chemical composition of molybdic ochre. *U. S. Geol. Survey Bull.* 490, 1911, pp. 84-82; *Am. Jour. Sci.*, ser. 4, vol. 23, April, 1907, pp. 297-303.
- Notes on powellite. *U. S. Geol. Survey Bull.* 490, 1911, pp. 80-83.
- SCHLIER, KARL. Über ein Molybdänbluerz-Vorkommen in Ober-Bayern. *Oesterr. Ztschr. Berg. und Hüttenwesen*, vol. 59, 1911, pp. 475-478. Describes an extensive wulfenite deposit in Upper Bavaria.
- SCHRADER, F. C., and HILL, J. M. Some occurrences of molybdenite in the Santa Rita and Patagonia mountains, Arizona. *U. S. Geol. Surv. Bull.* 430, 1910, pp. 154-163. Describes deposits at Helvetia and in Madera and Providencia canyons, also at Duquesne and in San Antonio Canyon.
- SMITH, G. O. A molybdenite deposit in eastern Maine. *U. S. Geol. Surv. Bull.* 260, 1905, pp. 197-199. Describes deposit at Cooper, Washington County.
- STEINHART, O. J. Metals and their ferroalloys used in the manufacture of alloy steels. *Trans. Inst. Min. and Met.*, vol. 15, 1905-1906, pp. 247-249. Describes ores and metallurgy of molybdenum.
- SWEZEY, R. O. Molybdenite deposit at Turn Back Lake, Quebec. *Can. Min. Jour.*, vol. 34, 1913, pp. 190-191. Describes occurrence and exploration work.



- SWINDEN, THOMAS. A study of the constitution of the carbon-molybdenum steels, with an appendix on the mechanical properties of some low molybdenum-alloy steels. Carnegie Scholarship Memoirs, Iron and Steel Inst. (London), vol. 5, 1913, pp. 100-168. A study of the recalescence curves, microstructure, and hardness of certain molybdenum steels as bearing on their constitution.
- . Carbon molybdenum steels. Carnegie Scholarship Memoirs, Iron and Steel Inst. (London), vol. 3, 1911, pp. 66-124. A study of the mechanical properties and electrical resistivity of certain molybdenum steels. A comparison with corresponding tungsten steels is of interest.
- TAYLOR, F. W. On the art of cutting metals. Trans. Am. Soc. Mech. Eng., vol. 28, 1907, pp. 31-350. Discusses molybdenum as a substitute for tungsten in high-speed tools, pp. 243-244.
- TREADWELL, W. D. Ueber die elektroanalytische Trennung des Kupfers von Wolfram und Molybdän. Ztschr. Elektrochem., vol. 19, 1913, pp. 219-221.
- VIGOUROUX, E. Sur les ferromolybdènes purs. Compt. rend., vol. 142, 1906, pp. 889-891, 928-930. Describes various iron-molybdenum compounds.
- VON LIPIN, W. Einige Eigenschaften des Molybdänstahl. Stahl und Eisen, 1897, vol. 14, pp. 571-572. Describes series of comparative tests on molybdenum and tungsten steels.
- WALKER, T. L. On the molybdenum ores of Ontario and British Columbia, 1911, pp. 65-66. Summary report of the mines branch for the calendar year ended Dec. 31, 1910.
- . Report on the molybdenum ores of Canada. Canadian Dept. of Mines, Mine Branch, Ottawa, 1911, No. 93, 64 pp.
- WEINIG, A. J. Molybdenum in cyanide solutions. Eng. and Min. Jour., vol. 97, 1914, p. 773. Describes qualitative test for molybdenum with potassium sulphocyanide.
- WELLS, J. W. Molybdenite—its occurrence, concentration, and uses. Canadian Min. Rev., vol. 22, 1903, pp. 113-118. Gives results of concentration tests and describes occurrences and uses in Canada.
- WINNE, R. Small electric furnace with heating element of ductile molybdenum. Trans. Am. Electrochem. Soc., vol. 20, 1911, pp. 287-292. Describes crucible and tube furnace, in which wire is wound on the alundum and hydrogen is used to protect the metal.
- WOOD, H. E. Concentration of molybdenite ores. Eng. and Min. Jour., vol. 93, 1912, pp. 227-228. Describes Wood flotation apparatus and its application to molybdenite ores.
- . The Wood flotation process. Trans. Am. Inst. Min. Eng., vol. 44, 1912, pp. 684-701. Good description of Wood process and apparatus, with results of tests of various ores, including molybdenite.
- 40167°—Bull. 111—16—9

## PUBLICATIONS ON MINERAL TECHNOLOGY.

A limited supply of the following publications of the Bureau of Mines has been printed, and is available for free distribution until the edition is exhausted. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

The Bureau of Mines issues a list showing all its publications available for free distribution as well as those obtainable only from the Superintendent of Documents, Government Printing Office, on payment of the price of printing. Interested persons should apply to the Director, Bureau of Mines, for a copy of the latest lists.

BULLETIN 16. The uses of peat for fuel and other purposes, by C. A. Davis. 1911. 214 pp., 1 pl., 1 fig.

BULLETIN 45. Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 64. The titaniferous iron ores of the United States, their composition and economic value, by J. T. Singewald, jr. 1913. 145 pp., 16 pls., 3 figs.

BULLETIN 81. The smelting of copper ores in the electric furnace, by D. A. Lyon and R. M. Keeney. 1915. 80 pp., 6 figs.

BULLETIN 84. Metallurgical smoke, by C. H. Fulton. 1915. 94 pp., 6 pls., 15 figs.

BULLETIN 85. Analyses of mine and car samples of coal collected in the fiscal years 1911 to 1913, by A. C. Fieldner, H. I. Smith, A. H. Fay, and Samuel Sanford. 1914. 444 pp., 2 figs.

BULLETIN 92. Feldspars of the New England and Northern Appalachian States, by A. S. Watts. 1915. 181 pp., 8 pls., 22 figs.

BULLETIN 100. Manufacture and uses of alloy steels, by H. D. Hibbard. 1915. 78 pp.

BULLETIN 106. The technology of marble quarrying, by Oliver Bowles. 1916. 174 pp., 12 pls., 23 figs.

TECHNICAL PAPER 8. Methods of analyzing coal and coke, by F. M. Stanton and A. C. Fieldner. 1913. 42 pp., 12 figs.

TECHNICAL PAPER 14. Apparatus for gas-analysis laboratories at coal mines, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 7 figs.

TECHNICAL PAPER 32. The cementing process of excluding water from oil wells, as practiced in California, by Ralph Arnold and V. R. Garfias. 1912. 12 pp., 1 fig.

TECHNICAL PAPER 38. Wastes in the production and utilization of natural gas, and means for their prevention, by Ralph Arnold and F. G. Clapp. 1913. 29 pp.

TECHNICAL PAPER 39. The inflammable gases in mine air, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 2 figs.

TECHNICAL PAPER 43. The influence of inert gases on inflammable gaseous mixtures, by J. K. Clement. 1913. 24 pp., 1 pl., 8 figs.

TECHNICAL PAPER 50. Metallurgical coke, by A. W. Belden. 1913. 48 pp., 1 pl., 23 figs.

TECHNICAL PAPER 66. Mud-laden fluid applied to well drilling, by J. A. Pollard and A. G. Heggem. 1914. 21 pp., 12 figs.

TECHNICAL PAPER 68. Drilling wells in Oklahoma by the mud-laden fluid method, by A. G. Heggem and J. A. Pollard. 1914. 27 pp., 5 figs.

TECHNICAL PAPER 70. Methods of oil recovery in California, by Ralph Arnold and V. R. Garfias. 1914. 57 pp., 7 figs.

TECHNICAL PAPER 76. Notes on the sampling and analysis of coal, by A. C. Fieldner. 1914. 59 pp., 6 figs.

TECHNICAL PAPER 88. The radium-uranium ratio in carnotites, by S. C. Lind and C. F. Whittemore. 1915. 29 pp., 1 pl., 4 figs.

TECHNICAL PAPER 95. Mining and milling of lead and zinc ores in the Wisconsin district, Wisconsin, by C. A. Wright. 1915. 39 pp., 2 pls., 5 figs.

TECHNICAL PAPER 110. Monazite, thorium, and mesothorium, by K. L. Kithil, 1915. 32 pp., 1 fig.

TECHNICAL PAPER 111. Safety in stone quarrying, by Oliver Bowles. 1915. 48 pp., 5 pls., 4 figs.

TECHNICAL PAPER 126. The casting of clay wares, by T. C. McDougal. 1916. 26 pp., 6 figs.

TECHNICAL PAPER 128. Quarry accidents in the United States during the calendar year 1914, compiled by A. H. Fay. 1915. 45 pp.

#### **PUBLICATIONS THAT MAY BE OBTAINED ONLY THROUGH THE SUPERINTENDENT OF DOCUMENTS.**

The editions for free distribution of the following Bureau of Mines publications are exhausted, but copies may be obtained by purchase from the Superintendent of Documents, Government Printing Office, Washington, D. C., or can be consulted at public libraries. Prepayment of the price is required and should be made in cash (exact amount) or by postal or express money order payable to the Superintendent of Documents.

The Superintendent of Documents is an official of the Government Printing Office and is not connected with the Bureau of Mines.

BULLETIN 3. The coke industry of the United States as related to the foundry, by Richard Moldenke. 1910. 32 pp. 5 cents.

BULLETIN 11. The purchase of coal by the Government under specifications, with analyses of coal delivered for the fiscal year 1908-9, by G. S. Pope. 80 pp. 10 cents. Reprint of United States Geological Survey Bulletin 428.

BULLETIN 12. Apparatus and methods for the sampling and analysis of furnace gases, by J. C. W. Frazer and E. J. Hoffman. 1911. 22 pp., 6 figs. 5 cents.

BULLETIN 19. Physical and chemical properties of the petroleum of the San Joaquin Valley, Cal., by I. C. Allen and W. A. Jacobs, with a chapter on analyses of natural gas from the southern California oil fields, by G. A. Burrell. 1911. 60 pp., 2 pls., 10 figs. 10 cents.

BULLETIN 26. Notes on explosive mine gases and dusts, with especial reference to explosions in the Monongah, Darr, and Naomi coal mines, by R. T. Chamberlin. 383 pp., 1 fig. 10 cents. Reprint of United States Geological Survey Bulletin 383.

BULLETIN 29. The effect of oxygen in coal, by David White. 80 pp., 3 pls. 20 cents. Reprint of United States Geological Survey Bulletin 382.

BULLETIN 38. The origin of coal, by David White and Reinhardt Thiessen, with a chapter on the formation of peat, by C. A. Davis. 1913. 390 pp., 54 pls. 80 cents.

BULLETIN 41. Government coal purchases under specifications, with analyses for the fiscal year 1909-10, by G. S. Pope, with a chapter on the fuel-inspection laboratory of the Bureau of Mines, by J. D. Davis. 1912. 97 pp., 3 pls., 9 figs. 15 cents.

BULLETIN 42. The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs. 20 cents.

BULLETIN 43. Comparative fuel values of gasoline and denatured alcohol in internal-combustion engines, by R. M. Strong and Lauson Stone. 1912. 243 pp., 3 pls., 32 figs. 20 cents.

BULLETIN 47. Notes on mineral wastes, by C. L. Parsons. 1912. 44 pp. 5 cents.

BULLETIN 63. Sampling coal deliveries and types of Government specifications for the purchase of coal, by G. S. Pope. 1913. 68 pp., 4 pls., 3 figs. 10 cents.

BULLETIN 70. A preliminary report on uranium, radium, and vanadium, by R. B. Moore and K. L. Kithil. 1913. 100 pp., 2 pls., 2 figs. 15 cents.

BULLETIN 73. Brass furnace practice in the United States, by H. W. Gillett. 1914. 298 pp., 2 pls., 23 figs. 45 cents.

BULLETIN 88. The condensation of gasoline from natural gas, by George A. Burrell, Frank M. Seibert, and G. G. Oberfell. 1915. 106 pp., 6 pls., 18 figs. 15 cents.

BULLETIN 100. Manufacture and uses of alloy steels, by H. D. Hibbard. 1915. 77 pp. 10 cents.

BULLETIN 104. Extraction and recovery of radium, uranium, and vanadium from carnotite, by C. L. Parsons, R. B. Moore, S. C. Lind, and O. C. Schaefer. 1915. 124 pp., 14 pls., 9 figs. 25 cents.

TECHNICAL PAPER 1. The sampling of coal in the mine, by J. A. Holmes. 1911. 18 pp., 1 fig. 5 cents.

TECHNICAL PAPER 3. Specifications for the purchase of fuel oil for the Government, with directions for sampling oil and natural gas, by I. C. Allen. 1911. 13 pp. 5 cents.

TECHNICAL PAPER 23. Ignition of mine gas by miniature electric lamps with tungsten filaments, by H. H. Clark. 1912. 5 pp. 5 cents.

TECHNICAL PAPER 26. Methods for the determination of the sulphur content of fuels, especially petroleum products, by I. C. Allen and I. W. Robertson. 1912. 13 pp., 1 fig. 5 cents.

TECHNICAL PAPER 27. Monthly statement of coal-mine accidents in the United States, January to August, 1912, and statistics for 1910 and 1911, compiled by F. W. Horton. 1912. 24 pp. 5 cents.

TECHNICAL PAPER 60. The approximate melting points of some commercial copper alloys, by H. W. Gillett and A. B. Norton. 1913. 10 pp., 1 fig. 5 cents.

TECHNICAL PAPER 90. Metallurgical treatment of the low-grade and complex ores of Utah, a preliminary report, by D. A. Lyon, R. H. Bradford, S. S. Arentz, O. C. Ralston, and C. L. Larson. 1915. 40 pp. 5 cents.

TECHNICAL PAPER 109. Composition of the natural gas used in 25 cities, with a discussion of the properties of natural gas, by G. A. Burrell and G. G. Oberfell. 1915. 22 pp. 5 cents.



# INDEX.

A.	Page.		Page.
Accidental mine, Ariz., wulfenite in.....	52	Canada, molybdenite in, production of.....	34
Achrematite, composition of.....	7, 14	molybdenite from, results of concentra-	
occurrence of.....	14	tion of.....	104
tests for.....	14	Carpenter, —, on properties of molybdenum	
Alaska, molybdenite in.....	86	steel.....	23
results of concentrating.....	104	Catherine's Hill, Me., occurrence of knightite	
Alloy steels, molybdenum in, value of....	20, 21, 25	at.....	17
uses of.....	20	Chemical reagents, manufacture of, use of	
<i>See also</i> Alloy steels named.		molybdenum in.....	20
Ammonia, production of, use of molybdenum		Chico, Mont., molybdenite near.....	73, 74
in.....	28	view of.....	64
Ammonium molybdate, use of.....	28	Chihuahua, Mexico, achrematite in.....	14
Andrews, E. C., on molybdenum in New		Climax, Colo., molybdenite near.....	63, 68, 69
South Wales.....	31	Coblentz, W. W., determination of emissivity	
Arizona, molybdenum ores in.....	45, 52-58, 86	of molybdenum by.....	18
<i>See also</i> Localities and mines named.		Colett, E., on determination of molybdenum..	43
Armor plate, use of molybdenum in.....	26	Collins mine, Ariz., buildings of, view of....	46
Arsenic in molybdenum ore, effect of.....	37	description of.....	46, 48
Aspen, Colo., molybdenite near.....	71	situation of, map showing.....	46
Austria, molybdenum ores in.....	14, 16	Concentration of ores, tests of.....	91, 92
		wulfenite in.....	46, 47
B.		Colorado, molybdenum ore in.....	45, 63-72, 86, 89
Baringer Hill, Tex., powellite at.....	14	map showing.....	66
Belonesite, composition of.....	7, 13	view of.....	66
occurrence of.....	13	<i>See also</i> localities and mines named.	
tests for.....	13	Colorado bureau of mines, acknowledgment to	6
Bennett mine, N. Mex., wulfenite in.....	77	<i>See also</i> Molybdenite; Wulfenite; and	
Benton mine, Ariz., molybdenite in.....	58	methods named.	
Bibliography.....	121-125	Concentrator, for water flotation.....	102, 103
Bismuth in molybdenum ore, effect of.....	37	results of tests with.....	98, 99
Blair, Thomas, on molybdenum in steel.....	21	Concunully, Wash., molybdenite near.....	85
Boericke, E. R., acknowledgment to.....	6	Connecticut, molybdenite in.....	86
Borchers, W., on acid-resisting steels.....	25	Cooke, Mont., molybdenite near.....	76
Boykin, R. O., production of molybdenum		Cooper, Me., production of molybdenum at..	32, 93
by.....	33, 50, 116	Copper in molybdenum ores, objections to... 37, 108	
Boykin & Hereford mill, Ariz., flow sheet of,		removal of.....	108, 109
figure showing.....	117	Copperville, Ariz., molybdenite near.....	52, 55
operation of.....	117, 118	ore from, production of.....	33, 105
tailing piles, view of.....	114	view of.....	52
view of.....	114	Cordite, use of molybdenum in.....	28
<i>See also</i> Mammoth, Ariz., mill at.		Corinthia, ilsemanite in.....	13
Breckenridge, Colo., molybdenite near.....	63, 67	Cornu, F., on occurrence of jordisite.....	9
Bromide, N. Mex., molybdenite at.....	77	Corona, Cal., molybdenite near.....	62
Buchanan, Paul, cited.....	62	ore from, view of.....	60
Bureau of Mines, determination of molybde-		Cotapaxi, Colo., molybdenite near.....	70
num by.....	44	Cripple Creek district, Colo., molybdenum	
investigation of.....	3, 4	minerals in.....	13, 16
Bureau of Standards, on melting point of		Crook A. R., on minerals associated with	
molybdenum.....	17	molybdenite.....	9
Burgess, G. K., determination of emissivity of		Crown Point mine, Wash., description of....	79-83
molybdenum by.....	18	molybdenite in, cost of mining.....	82
C.		view of.....	8, 80
Cahen, Edward, on occurrence of jordisite...	9	production of.....	32, 33, 82
Caliente, Cal., molybdenite near.....	63	situation of, map showing.....	80
California, molybdenum ore in.....	45, 58-63, 86, 89	Cutter, N. Mex., production of wulfenite	
analysis of.....	12	from.....	34, 78
<i>See also</i> localities and mines named.			

D.	Page.		Page.
Daily, G. H., acknowledgment to.....	6	Guillet, Leon, on ratio of molybdenum to tungsten in steel.....	22
Dana, E. S., acknowledgment to.....	6	H.	
on composition of molybdenum ores..	10, 13, 17	Hall, M. C., cited.....	61
Debray, —, on fusibility of molybdenum....	17	Hamburg mine, Ariz., wulfenite in.....	50
Defacqz, E., on specific heat of molybdenum..	18	Hasselberg, B., determination of emissivity of molybdenum by.....	19
Dentistry, use of molybdenum in.....	20, 27	Haynes, Elwood, on molybdenum in steel alloys.....	26
Des Cloizeaux, Alfred, on occurrence of molybdurane.....	16	Head, R. E., acknowledgment to.....	6
Disinfectants, use of molybdenum in.....	20	Helvetia, Ariz., molybdenite near.....	56, 57
Doerner, H. A., acknowledgment to.....	5	ore from, view of.....	52
Dome, Ariz., wulfenite near.....	50	Hereford, F. H., production of molybdenum by.....	33, 116
Duquesne, Ariz., molybdenite at.....	58	Hess, F. L., acknowledgment to.....	6
Dyes, use of molybdenum in.....	20	cited.....	62, 93
E.		Hill, J. M., on molybdenite in Arizona.....	56, 58
Eckhardt, M., on determination of molybdenum.....	43	Hills, B. W., cited.....	17
Electric furnaces, use of molybdenum in....	20, 27	Hillsboro, N. Mex., wulfenite at.....	78
Electric lamps, use of molybdenum in.....	27	Homestake, Mont., molybdenite from.....	32
Electrostatic methods of concentrations, apparatus for.....	96, 97	Hoover, T. J., on oil flotation processes.....	101, 105-107
costs of.....	100	Hortense, Colo, molybdenite from, analysis of..	12
factors governing.....	94, 96	I.	
limitations of.....	95	Idaho, molybdenite in.....	86
results of tests of.....	98-100	Idaho Springs, Colo., view of molybdenum mineral near.....	15
Elmore process, results of tests with.....	107, 108	Iddings, J. P., on molybdenite near Chico, Mont.....	73
Emigrant, Mont., molybdenite near.....	73	Ilsemanite, composition of.....	7, 12
Empire, Colo, molybdenite near.....	5, 63, 64-67	occurrence of.....	13
ore from, production of.....	33, 67	Iridium steel, analysis of.....	24
view of.....	60, 64	J.	
Eosite, composition of.....	7, 14, 15	Jacobson, R. C., analyses of molybdenite by.....	53
occurrence of.....	15	Jordisite, characteristics of.....	9
tests for.....	15	K.	
F.		Kellog, J. M., on wulfenite deposits.....	51
Farrington, O. C., acknowledgment to.....	6	Kelvin, Ariz., wulfenite near.....	51
Ferris, J. B., cited.....	63	Kithil, K. L., acknowledgment to.....	5
Ferromolybdenum, use of.....	26	Knight, O. W., discovery of knightite by....	17
Ferrotungsten, prices for.....	39	knightite, occurrence of.....	16, 17
Flotation processes, principles underlying... <i>See also</i> Elmore process, oil-flotation; water-flotation.	101	Kokomo, Colo, molybdenite near.....	72
Ford, W. E., acknowledgment to.....	6	L.	
Forsythe, W. E., determination of emissivity of molybdenum by.....	19	Lamy, L., on prices for molybdenite concentrates.....	38
France, demand for molybdenum in.....	35	Laney, F. B., acknowledgment to.....	6
Frenzel, A. B., acknowledgment to.....	6	on molybdenite deposits.....	15, 83
Fusion method for determination of molybdenum.....	41	Lawrence, H. F., on molybdenite in Montana.....	74
G.		Laws, Cal., molybdenite near.....	61
Germany, as market for molybdenum.....	35	Leader mine, Ariz., molybdenite in.....	56, 57
Giesen, Walter, on ratio of molybdenum to tungsten in steel.....	22	ore from, view of.....	52
Gin, Gustave, on composition of molybdenum minerals.....	14, 16	Leal, H., acknowledgment to.....	6
Glazes, use of molybdenum in.....	20	Leviathan mines, Ariz., description of.....	52-54
Gleason, Ariz., wulfenite near.....	51	molybdenite from, concentration of.....	54
Gledhill, J. M., on properties of molybdenum steel.....	23, 24	Lindgren, Waldemar, cited.....	13, 16
Gold in molybdenum ore, occurrence of.....	11, 115	Lively, C. S., acknowledgment to.....	6
recovery of.....	115	Loomis, Wash., molybdenite near.....	83
Gratacap, L. P., acknowledgment to.....	6	workings of, view of.....	80
Great Britain, as market for molybdenum...	35, 39	Lordsburg, N. Mex., wulfenite near.....	78
Guichard M., on specific heat of molybdenum..	18	Low, A. H., cited.....	41
		Lucky Bill mine, N. Mex., wulfenite in.....	78
		ore from, view of.....	10

M.	Page.		Page.
Magdalena, N. Mex., molybdenite at.....	77	Molybdate, view of.....	10
Magnet steel, composition of.....	25	Molybdoferriite, composition of.....	16
Maine, molybdenite in.....	86	Molybdurane, composition of.....	16
Mammoth, Ariz., wulfenite mill at, concen- trates from.....	33, 118	Montana, molybdenite in.....	45, 72-77
operations of.....	5, 115-119	Moore, R. B., acknowledgment to.....	5
view of.....	66	Morgan, J. C., acknowledgment to.....	5, 41
<i>See also</i> Boykin & Hereford mill.		Myers, W. S., acknowledgment to.....	6
Mammoth mine, Ariz., buildings of, view of... description of.....	46 46-48	N.	
situation of, map showing.....	46	Nathrop, Colo., molybdenite near.....	70
wulfenite ore from, production of.....	32	Nevada, molybdenum ores in.....	87, 89
view of.....	8	New Hampshire, molybdenum ores in.....	87
Marble, Colo., molybdenite near.....	71	analysis of.....	12
Massachusetts, molybdenum ores in.....	86, 89	New Jersey, molybdenite in.....	87
Melissa mine, Ariz., wulfenite in.....	50	New Mexico, molybdenum ores in.....	45, 77-79
Mendenhall, C. E., determination of emis- sivity of molybdenum by.....	18	New South Wales, molybdenum in, produc- tion of.....	5, 29, 31
Merrill, G. D., acknowledgment to.....	6	New York, molybdenum ores in.....	87, 89
Metallic molybdenum, prices for.....	39	Nogales, Ariz., molybdenite near.....	57
uses of.....	27	North Carolina, molybdenite in.....	88
Meyer, A. R., cited.....	17	Northrup, E. F., cited.....	28
Mickle, K. A., on flotation processes.....	101	Norway, molybdenum in, production of.... 29, 31, 32, 107	5,
Minnesota, molybdenite in.....	86	O.	
Modoc mine, N. Mex., molybdenite at.....	77	Oak Springs, Nev., powellite near.....	14
Mohawk mine, Ariz., wulfenite in.....	51	Oil flotation, costs of.....	107
Moissan, Henri, cited.....	16	treatment of molybdenite ores by.....	106
melting of molybdenum by.....	17	Oklahoma, molybdenite in.....	88
on properties of molybdenum.....	17, 20	Old Yuma mine, Ariz., description of.....	48-50
Molybdenite, composition of.....	7	installation of mill at.....	5
concentration of.....	91-93, 97-100	production of molybdenum at.....	34, 111, 112
results of.....	107, 108	Ophir, Mont., molybdenite near.....	69, 75
minerals associated with.....	9, 53, 55, 65, 67, 68, 70-73, 75, 76, 78, 81, 83, 84, 86-88, 95	ore from, view of.....	74
mining of, costs of.....	54	Oregon, molybdenite in.....	88
occurrence of.....	8, 9, 58-76, 79-88	Organ, N. Mex., molybdenum deposits at....	77
prices for, discussion of.....	38, 39	Oroville, Wash., molybdenite near.....	85
properties of.....	7, 101	Oscillating table, concentration of wulfenite ores on.....	112-114
tests for.....	8	P.	
views of.....	8, 52, 60, 64, 66	Parkdale, Colo., molybdenite near.....	70
Molybdenum, characteristics of.....	17	Parsons, C. I., acknowledgment to.....	5
deposits of. <i>See</i> States and localities named.		Patagonia, Ariz., molybdenum ores near.....	58
determination of.....	43	Pateraite, composition of.....	7, 14
ductile, electrical resistance of.....	18	Pennsylvania, molybdenum ores in.....	88
duty on.....	34	Phoenixville, Pa., wulfenite from, analysis of.	10
in steel, ratio of, to tungsten.....	22	Pirani, M. V., cited.....	17
relative advantages of.....	5	Plumb, A. M., acknowledgment to.....	6
market for, factors governing.....	3, 4, 35-37, 39	Pohnsdorf, A. G., acknowledgment to.....	6
melting point of.....	17	Porvenir, N. Mex., molybdenite near.....	77-79
precipitation of, methods for.....	42	ore from, production of.....	33
production of, in United States.....	32, 33, 36	view of.....	74
properties of.....	17-20	Powellite, composition of.....	7, 13
recovery of, methods for.....	91-100	occurrence of.....	14
selling price of.....	4	tests for.....	14
solution of, methods for.....	41, 42	Powers, C. R., on wulfenite deposits.....	51
tests for.....	40	Pratt, J. H., cited.....	32, 51
weighing of.....	43	Pulsator jig, concentration of wulfenite ores on.....	111-114
<i>See also</i> Metallic molybdenum and molyb- denum minerals named.		Pyrite in molybdenum concentrates, objec- tions to.....	108
Molybdenum indigo, use of.....	28	removal of.....	108, 109
Molybdenum tannate, use of.....	28	Pyrrhotite in molybdenum concentrates, ob- jections to.....	108
Molybdenum wire, tensile strength of.....	18		
Molybdate, composition of.....	7, 11, 12		
occurrence of.....	11, 12		
tests for.....	12		

Q.	Page.	T.	Page.
Queensland, production of molybdenum in . . .	5, 29, 30	Tailing piles, recovery of wulfenite from . . .	116
		view of . . .	66, 114
R.		Telluride, Colo., molybdenite from, analy-	
Radersburg, Mont., wulfenite near . . .	72	sis of . . .	12
Randall, D. L., on volumetric determination		Texas, molybdenite in . . .	88
of molybdenum . . .	43	Tonasket, Wash., molybdenite near . . .	83
Ransome, F. L., cited . . .	13, 16	Tool steel, composition of . . .	24
Red Cloud mine, Ariz., wulfenite in . . .	50	cutting speed of . . .	24
Red Mountain, Colo., molybdenite in . . .	5, 63-67	molybdenum in . . .	23
ore from, production of . . .	33	See also Iridium steel.	
view of . . .	64	Troy, Ariz., wulfenite near . . .	51
Rhode Island, molybdenite in . . .	88	Tucson, Ariz., molybdenum ores near . . .	48, 50, 57
Ridley mine, Ariz., molybdenite in . . .	57	concentration of . . .	5, 34, 111, 112
Risley, J. M., on molybdenite in Washington .	84, 85	Tungsten in molybdenum ore, effect of . . .	37
Robinson, Colo., molybdenite near . . .	68-70	in steel, ratio of molybdenum to . . .	22
Robischaud, P., on molybdenite deposit in		relative advantages of . . .	5
Washington . . .	84	selling price of . . .	4
Rociada, N. Mex., molybdenite at . . .	77	Tungsten wire, tensile strength of . . .	18
Rolling and screening, concentration by . . .	93, 94	Twin Bridges, Mont., wulfenite near . . .	72, 76, 77
Romero mine, N. Mex., molybdenite in . . .	78, 79		
ore from, view of . . .	74	U.	
Roscoe, H. E., on chemical properties of mo-		Umpleby, J. B., on minerals associated with	
lybdenum . . .	20	molybdenite . . .	9
S.		United States, molybdenum in, imports of . .	34, 35
Sackett, E. C., on molybdenite near Chico,		occurrence of . . .	5, 86-90
Mont . . .	74	production of . . .	3, 4, 32-34
St. Cloud, Colo., molybdenite near . . .	72	See also States and localities named.	
San Diego, Cal., molybdenite near . . .	60, 61	United States Geological Survey, on produc-	
ore from, concentration tests of . . .	98	tion of molybdenum . . .	32
Santa Maria mine, Cal., molybdenum ore		Utah, molybdenum ores in . . .	88-90
from, view of . . .	8, 10, 60		
Santa Rita, N. Mex., wulfenite near . . .	78	V.	
ore from, view of . . .	10	Vail, Ariz., molybdenite at . . .	57
Saxony, occurrence of jordisite in . . .	9	Vanadium, use of, in tool steel . . .	24
Schaller, W. T., cited . . .	11-13	Vermont, molybdenite in . . .	88
Scheelite, occurrence of molybdenum in . . .	17	Vulture mine, Ariz., wulfenite deposits in . .	52
Schorlemmer, C., on chemical properties of			
molybdenum . . .	20	W.	
Schrader, F. C., on molybdenite in Arizona .	56, 58	Waltenberg, R. G., determination of emis-	
Schultz, Ariz., mine at, production of molyb-		sivity of molybdenum by . . .	18
denum from . . .	32	Warren, C. H., acknowledgment to . . .	6
view of . . .	46	Washington, molybdenite in . . .	45, 79-85
Scotland, occurrence of eosite in . . .	15	Water flotation, factors governing . . .	102
Shepard, F. E., acknowledgment to . . .	6	results of tests of . . .	103-105
Skykomish, Wash., molybdenite near . . .	85	Weed, W. H., on molybdenite in Montana .	73
Snyder, Ira, on molybdenite deposit in Colo-		Westcliffe, Colo., molybdenite near . . .	71
rado . . .	69	Widmar, R. J. A., acknowledgment to . . .	6
Sodium molybdate, use of . . .	28	Wire, molybdenum, uses of . . .	27
South Carolina, molybdenite in . . .	88	Wolff, J. E., acknowledgment to . . .	6
South Dakota, wulfenite in . . .	89	Wood, H. E., acknowledgment to . . .	6
South Hecla mine, Mich., powellite at . . .	14	on water flotation . . .	103, 104, 109
Steel, molybdenum, early objections to . . .	21	Wootton, W. O., on occurrence of jordisite . .	9
effect of . . .	22, 25	Wulfenite, composition of . . .	7, 9, 10, 118
investigations of . . .	21, 22	concentration of . . .	92, 110-112, 114
manufacture of . . .	22	minerals associated with . . .	11,
See also Alloy steels; magnet steel; tool		45, 46, 47, 49, 50, 76, 77, 89, 90	
steels.		occurrence of . . .	10, 45-52, 58, 63, 76, 77, 89, 90
Steel wire, tensile strength of . . .	18	production of, in United States . . .	32, 33, 36
Stellite, analysis of . . .	27	tests for . . .	10
use of molybdenum in . . .	26	views of . . .	8, 10, 46
Stoughton, Bradley, on composition of armor		Wulfenite concentrates, prices for . . .	38, 39
plate . . .	26	screen test of . . .	119
Sullivan, J. J., wulfenite sampled by . . .	51	Wyoming, molybdenite in . . .	88
Swart, W. G., acknowledgment to . . .	6		
Swinden, Thomas, investigations of molybde-		X.	
num steel by . . .	21	X-ray tubes, use of molybdenum in . . .	20
		Y.	
		Young, William, acknowledgment to . . .	6



DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

MINING AND PREPARING  
DOMESTIC GRAPHITE FOR CRUCIBLE USE

BY

GEORGE D. DUB and FREDERICK G. MOSES

WITH A

CHAPTER ON METHODS OF ANALYSIS  
USED BY THE BUREAU OF MINES

BY

G. B. TAYLOR and W. A. SELVIG



WASHINGTON  
GOVERNMENT PRINTING OFFICE

1920

The Bureau of Mines, in carrying out one of the provisions of its organic act—to disseminate information concerning investigations made—prints a limited free edition of each of its publications.

When this edition is exhausted copies may be obtained at cost price only through the Superintendent of Documents, Government Printing Office, Washington, D. C.

The Superintendent of Documents *is not an official of the Bureau of Mines*. His is an entirely separate office and he should be addressed:

SUPERINTENDENT OF DOCUMENTS,  
*Government Printing Office,*  
*Washington, D. C.*

The general law under which publications are distributed prohibits the giving of more than one copy of a publication to one person. The price of this publication is 20 cents.

*First edition. January, 1920.*

# CONTENTS.

	Page.
Mining and preparing domestic graphite for crucible use.....	7
Prefatory statement.....	7
Introduction.....	8
Acknowledgment.....	8
Literature.....	8
Development of graphite industry in the United States.....	9
Establishment of permanent domestic industry.....	10
Graphite industry in Alabama.....	11
Plants.....	11
Ore mined and methods of mining.....	11
Crushing methods.....	13
Concentration methods.....	13
Water "skin flotation".....	14
Munro washer.....	16
Colmer washer.....	16
New Munro washer.....	17
Pneumatic process.....	17
Log-washer process.....	17
Oil-froth flotation systems.....	18
Minerals separation system.....	19
Callow pneumatic system.....	19
Simplex system.....	19
Refining methods.....	23
Air classification.....	24
Recovery of finished products.....	26
Comparative costs of erecting plants.....	27
Field sampling.....	27
Graphite industry in New York.....	27
Plants.....	27
Occurrence of ore.....	28
Mining methods.....	28
Open-pit mining.....	28
Underground mining.....	28
Concentrating methods.....	29
Refining methods.....	30
The industry in Pennsylvania.....	30
Plants.....	30
Occurrence of ore.....	30
Mining methods.....	32
Concentrating methods.....	32
Refining methods.....	32
The industry in Texas.....	33
Occurrence of ore.....	33
Mining methods.....	33
Concentrating methods.....	34
Recovery of graphite from "kish".....	35

Mining and preparing domestic graphite for crucible use—Continued.	Page.
Experimental work on concentration and refining	35
Experimental work on crucible manufacture	35
Photomicrography	36
Methods of sampling	36
Methods of analysis	38
Tentative specifications	38
Carbon content	40
Screen analysis	40
Present status of graphite mining	41
Method for rapid analysis of graphite used by the Bureau of Mines	43
Preparation of sample	44
Determination	44
Moisture	44
Volatile matter	44
Ash	44
Graphitic carbon	45
Graphitic carbon by combustion	45
Preliminary treatment of ores	45
Preliminary treatment of concentrates	45
Refining Alabama flake graphite for crucible use	46
Introduction	46
Requirements for crucible graphite	46
Present practice	46
Concentrates used in tests	47
Physical properties of graphite and associated gangue minerals	49
Aspirator tests	50
Tests with pneumatic jig	53
Tests with pebble mill	54
Discussion of pebble-mill results	58
Tests with electrostatic separator	58
Summary of results with electrostatic separator	64
Flotation as a finishing process	65
Flotation tests	65
The buhr mill; its place in graphite finishing	66
Relation of flake structure to finishing	67
Impurities in graphite	69
Results from combination of various treatments	70
General summary and discussion of results	73
Publications on mining and milling methods	75
Index	79

---

## TABLES.

---

TABLE 1. Results of screen analyses and assays of different samples selected for experiment	48
2. Physical properties of graphite and of associated minerals	49
3. Results of screen analyses and of assays of aspirator concentrates	51
4. Results of screen analyses and assays of jig concentrates and tailing	54



TABLE 5. Results of grinding graphite samples in pebble mill with equal parts of water.....	Page. 56
6. Results of grinding graphite sample 2 in pebble mill with different percentages of water.....	58

## ILLUSTRATIONS.

PLATE I. A, Improvised aspirator used in tests at Salt Lake City Station; B, Buhr mill used in grinding tests.....	50
II. A, Pneumatic jig used in tests; B, Buhr mill used in tests.....	53
III. Electrostatic graphite separator.....	60
IV. A, Microphotograph of graphite ore, showing irregular structure; B, Microphotograph of graphite ore, showing banded structure.....	66
V. A, Microphotograph of graphite ore, showing a large plate with a triangular inclusion; B, Microphotograph of graphite ore, showing impurities embedded in flakes.....	67
FIGURE 1. Flow sheet of water "skin flotation" system.....	14
2. Section through original Munro washer.....	16
3. Section through Colmer washer.....	16
4. Section of new Munro washer.....	17
5. Flow sheet of log-washer process.....	18
6. Flow sheet of Mineral separation system.....	20
7. Flow sheet of Callow pneumatic system.....	21
8. Flow sheet of simplex system.....	22
9. Section of simplex washer.....	23
10. Flow sheet of graphite refining plant.....	24
11. Two types of air classifiers.....	25
12. Flow sheet of New York concentrating plant.....	29
13. Contemplated flow sheet for Huff electrostatic system.....	31
14. Method recommended for sampling finished sacked product..	36
15. Sections of sampling device.....	38
16. Curves showing results of grinding sample 2 in pebble mill with equal parts of water.....	56
17. Curves showing results of grinding sample 4 in pebble mill with equal parts of water.....	57
18. Curves showing results of grading sample 2 in pebble mill with different percentages of water.....	57
19. Electrostatic graphite separator designed by Bureau of Mines.....	61
20. Wiring diagram of electrical equipment used for electrostatic graphite separator designed by Bureau of Mines....	62



# MINING AND PREPARING DOMESTIC GRAPHITE FOR CRUCIBLE USE.

---

By GEORGE D. DUB and FREDERICK G. MOSES.

---

## PREFATORY STATEMENT.

The Bureau of Mines, in connection with the investigations of war minerals it conducted, examined the graphite deposits of this country, studied the methods of mining and preparation used, and sought to devise more efficient methods for the preparation of domestic graphite, and to standardize the quality of the product. The graphite investigations covered three general phases, as follows:

1. An examination of the deposits in Alabama and other States, and a survey of the methods of mining and preparation used. In this connection methods of sampling and analysis were noted and experiments made to determine a standard method of sampling and a rapid but accurate method of analysis.

2. Experimental work on the concentration and refining of domestic crucible graphite to improve the quality of the product and to lessen waste.

3. Experimental work in crucible manufacture to determine the properties of domestic flake and the maximum proportions that might be used without impairing the quality of the crucibles. This work was accompanied by photomicrographic study of crucible structure.

In this bulletin are presented the results obtained in the first two phases of the work outlined above. The bulletin is in two parts. The first part describes the methods of mining and milling used, suggests a standard method of sampling finished graphite, and describes a rapid and convenient method of analysis developed at the Pittsburgh station of the Bureau of Mines, to which had been assigned the analytical and microscopic work. The second part describes experiments on the concentration and milling of graphite, which were made at the Salt Lake City, Utah, station. As soon as the reports incorporated in this bulletin were ready, mimeograph copies were distributed to those persons most interested in order to make the results immediately available.

The experimental work on crucible manufacture, assigned to the Columbus, Ohio, station, yielded results which were made known to the industry. A report on this work will be published later as a separate bulletin.

# MINING AND MILLING METHODS IN ALABAMA.

---

By GEORGE D. DUB.

---

## INTRODUCTION.

The writer, in connection with the graphite investigation of the Bureau of Mines, war minerals investigations, was charged with the examination of mining and milling methods. During the course of the field work, Alabama, New York, Pennsylvania, and Texas, the principal producing districts in the United States, were visited. In addition, the writer visited a plant in which graphite is refined from "kish," a by-product obtained from iron blast-furnaces and steel plants. The field work was planned to permit of studying the methods of mining and milling and of refining graphite for market. The results of this observation and study are set forth in the following pages. A standard method for sampling finished graphite is suggested, and a rapid, convenient method of analysis used by the bureau is described.

## ACKNOWLEDGMENT.

Acknowledgment is made of the whole-hearted cooperation of producers in all of the fields. The Graphite-Producers Association of Ashland, Ala., the members of the staff of the office of the State geologist of New York, and of the various bureaus and boards in Washington have been of great assistance in furthering the aims of this investigation and in making effective various recommendations made from time to time. To H. S. Mudd, mineral technologist of the Bureau of Mines, special credit is due for his painstaking review of the manuscript.

## LITERATURE.

A bibliography on graphite has been prepared by Ferguson <sup>a</sup>. The Canadian Bureau of Mines is revising Cirkel's monograph on graphite <sup>b</sup>, Prof. B. L. Miller is revising Cirkel's monograph on deposits of Pennsylvania <sup>c</sup> for the State geological survey, and the

---

<sup>a</sup> Ferguson, H. G., Graphite in 1917: Mineral Resources, U. S., 1917, U. S. Geological Survey, 1918, pp. 117-119.

<sup>b</sup> Cirkel, Fritz, Graphite; its properties, occurrence, refining and uses: Canada Department of Mines, Mines Branch, 1907, 307 pp.

<sup>c</sup> Miller, B. L., Graphite deposits of Pennsylvania: Topographic and Geol. Survey of Pennsylvania, 1912, 147 pp.



New York State deposits are fully described in a bulletin of the State Museum by H. L. Alling<sup>a</sup>. In view of the detailed geological discussions and the full descriptions of milling and refining apparatus and of methods in the publications mentioned, such phases receive only incidental mention in this report. No statistics except those not available in State or United States Geological Survey bulletins have been incorporated. The aim is simply to describe present practice and recent developments.

## DEVELOPMENT OF GRAPHITE INDUSTRY IN THE UNITED STATES.

Before 1915 nearly all the graphite used in crucible manufacture in this country was imported. Little domestic graphite was used in crucibles, that mined being used chiefly for the manufacture of lubricants, paint, foundry facings, and other purposes. In 1915 the demand for graphite crucibles increased greatly, because of the placing of large foreign contracts for munitions and ordnance with American plants, and because of the inferiority of crucibles made from clay other than that from Klingenberg, Bavaria, which had been cut off by the blockade. Certain crucible manufacturers, fearing a shortage of foreign graphite, offered high prices for domestic flake graphite. This stimulated development of the graphite-bearing rocks of Alabama. Also, the uncertainty regarding foreign supplies caused crucible manufacturers to use a larger proportion of domestic graphite in their mixtures than they had ever before attempted.

During 1916 and 1917 the amount of graphite imported was about eight times the domestic production. Imported graphite comes mainly from Ceylon, Madagascar, and Korea. The amorphous graphite from Korea can be easily replaced by that from Mexico, or, if necessary, by development of somewhat lower grade deposits in the United States. Madagascar and Ceylon have furnished about 70 per cent of the total graphite imported, and of this amount approximately 90 per cent has been used in crucible manufacture.

Until the United States entered the war in 1917, the use of Ceylon and Madagascar graphites by American manufacturers was unrestricted as long as guaranties could be furnished to the British and French Governments that the products into which these graphites entered would not fall into the hands of the enemy. On the declaration of war in April, 1917, these guaranties became unnecessary, as they were superseded by broader American defense measures.

During the interval in the winter of 1917-18 when freight conditions were most congested, an embargo against the shipment of

---

<sup>a</sup> Alling, H. L., The Adirondack Graphite Deposits, N. Y. State Museum Bull. 199, 1917, 150 pp.

domestic graphite was ordered and permitted to remain in force until the early part of March, 1918. The result of this embargo was a stagnation of the domestic mining industry. The removal of the freight restriction on domestic graphite was followed in April by a complete embargo on importations of overseas graphite during the interval April 15, 1918, to July 1, 1918. For the remainder of the calendar year 5,000 long tons was to have been permitted to enter this country. This order was modified to the extent of allowing importations during the period April 15, 1918, to July 1, 1918; the total amount of these imports, however, was to have been deducted from the 5,000 tons scheduled to enter during the last six months of the year.

Toward the end of June, 1918, in view of the necessity for conserving shipping for the direct military program and after stocks of overseas graphite in the hands of crucible makers, refiners, and dealers were found sufficient to last about six months, it was decided after July 2, 1918, to restrict completely the importation of overseas graphite for the rest of 1918. This order was followed on August 10, 1918, by a request from the War Industries Board that all crucible makers use 20 per cent domestic flake graphite in their crucible graphite mixtures for the rest of 1918 with an increase of 25 per cent for 1919. This request carried with it the statement that applications for import licenses of manufacturers not complying with the provisions of the request would not be approved by the War Industries Board.

The War Trade Board ruling of July 2, 1918, was superseded on October 17, 1918, by a ruling permitting imports of overseas graphite by manufacturers whose applications had been approved by the War Industries Board. The purpose of this ruling was to permit the carrying of 3 or 4 months' supply of graphite, in view of the length of time required for shipments from Ceylon and Madagascar to reach the factory in which they were to be used.

#### ESTABLISHMENT OF PERMANENT DOMESTIC INDUSTRY.

Before the declaration of war in 1914, all crucible makers in the United States, without exception, used clay imported from Klingenberg, Bavaria, in the manufacture of crucibles. Little work of any kind had been done with domestic or English clay. With the declaration of war the cutting off of Bavarian clay forced the crucible maker to turn his attentions to other clays. Although at first serious difficulties were encountered, the clay problem is fairly well in hand with the result that crucible efficiency compares favorably in service with prewar standards. After clay difficulties had been largely surmounted, the problems of crucible manufacture were considered solved. In the course of experiments, no great success at-

tended the use of more than 25 per cent domestic flake graphite in crucible graphite mixtures. It was, therefore, fitting that the Bureau of Mines should thoroughly investigate not only the use of domestic flake in crucible manufacture, but also the methods of manufacturing crucibles from such graphite.

The future of the graphite industry and the effect of improvements and changes in the brass and steel industries have been well outlined by Ferguson.<sup>a</sup> If new and increasing uses can be developed for graphite, it may be possible through the manufacture of graphite articles of commerce in the various mining districts to establish the domestic mining industry on a firmer basis than is possible with many small operating units that produce only crucible stock. The by-products, No. 2 flake and graphite dust, which are drugs on the present market, would then be of value. The graphite-mining industry made a hearty response to the market for crucible grades created by the request of the War Industries Board, and it would be unfortunate if, with the reestablishment of peace, this industry should revert to its prewar basis.

## GRAPHITE INDUSTRY IN ALABAMA.

### PLANTS.

There are 39 plants in the three graphite-producing counties of Alabama: Clay, Coosa, and Chilton. About October 1, 1918, four of these plants were in course of erection and fourteen were temporarily closed because of changes in method of treatment or because of plant fires. A little over half of the plants were then operating, and only six were running full time. After the cessation of hostilities production fell rapidly, and in June, 1919, probably not more than half a dozen plants were in operation.

As this industry developed chiefly during 1917 and 1918, operating difficulties still occur.

The plants enumerated have actual operating capacities ranging from 4 to 15 tons an hour, but the rated capacities are about double these figures.

### ORE MINED AND METHODS OF MINING.

The graphite occurs in the Talladega schist and is accompanied by quartz, feldspar, mica, other accessory minerals, and alteration products. Only the upper 30 to 60 feet of the graphite schist is mined. This consists of decomposed and weathered schist and is normally rather soft and easily broken, while the unweathered schist or

<sup>a</sup> Ferguson, H. G., Graphite in 1917: Mineral Resources U. S., 1917, U. S. Geol. Survey, 1918, pp. 97-119.



"blue rock" is comparatively hard and contains considerable pyrite. For this reason, operators are loath to work the "blue rock." Alabama ore contains, as far as is known, no amorphous graphite.

The ore is covered with 1 to 6 feet of overburden. This carries some graphite, but contains so much clay and vegetable matter that it is removed and dumped as waste to obviate milling difficulties caused by the presence of these materials. The stripping is done with plows, scrapers, and wheelbarrows, and does not involve much labor or expense.

All of the companies have open-pit workings. Machine drills are employed at only one mine, where self-rotating hammer drills of the jackhammer type are used to drill holes 15 feet in depth. All other drilling is done by means of a "jumper" drill, holes being drilled 8 to 30 feet deep. All blasting is done with black powder. The deeper holes are chambered with dynamite or blasting gelatin.

The broken ore is loaded into cars by hand, or by steam shovels with a dipper capacity of five-eighths of a cubic yard. With hand loading, end-dump or side-dump cars of 1 to  $1\frac{3}{4}$  ton capacity are employed. With steam shovels, side-dump contractors' cars of 4 to 5 ton capacity are used. About one-fourth of the plants in this district are so situated as to necessitate hoisting ore out of the open pit to the mill. At the plants in which the crude ore enters the mill by gravity, hand tramming is employed, except where the open pit is situated at a considerable elevation above the mill; in such plants, boxed chutes with hand tramming or gravity planes are used. Balanced hoisting is used in only three or four plants. At one plant, a narrow-gage railroad transports ore from the open pit to the mill.

The absence of enough storage room at the head of the mill or at some point between the open cut and the mill is practically general in the district. Only a small proportion of the plants have storage capacity sufficient to permit the accumulation of enough ore to run one shift of 10 hours, the majority of plants having storage capacity to enable milling to continue only for a half shift. Mining is carried on in the open and is dependent, to a certain degree, on the weather. Shut-downs in the mills, owing to operating troubles, are frequent. The result, therefore, of the lack of storage is a loss of operating time, the elimination of which would soon pay for the installation of adequate storage. This point can not be too strongly urged for these small, and, mostly, single-unit plants.

The ore in Alabama averages about  $2\frac{1}{2}$  per cent graphitic carbon. From this is obtained generally three products—No. 1 crucible flake, No. 2 flake, and dust. The aim in concentrating is to recover as much as possible of the No. 1 flake, analyzing 85 per cent graphitic carbon and remaining on a No. 8 silk cloth of 86 mesh. No. 2 flake, analyz-



ing 75 to 80 per cent graphitic carbon and of finer size than the No. 1 flake, is a by-product grade for which the market is limited. The dust analyzes 30 per cent or more graphitic carbon and is a greater drug on the market than the No. 2 flake. Hence, the problem of ore dressing is complicated by the necessity of recovering as much as possible of the graphite in the ore in such form as to be marketable as No. 1 flake. The measure of efficiency, therefore, in graphite milling is not the percentage of recovery but the number of pounds of No. 1 flake recovered per ton of ore.

Unfortunately graphite plants do not make a practice of analyzing their mill heads. It is impossible therefore, to establish a relation between the total content of graphite in the ore and the amount recovered as No. 1 flake. Average figures as to the actual production per ton of ore are stated on page 27.

The by-product grades can readily be used in the manufacture of lubricating flake, paint stock, foundry facings, stove polish, etc., but the demands of plants manufacturing these products are not large and the market does not absorb all the No. 2 flake and dust produced.

#### CRUSHING METHODS.

Many different crushing machines and combinations are used in Alabama in the dressing of graphite ore. The coarse crusher most commonly used is a jaw crusher of the Blake or Dodge type that reduces the ore to about  $1\frac{1}{2}$  inch size. In some plants, a toothed impact roll is used for coarse crushing. In plants where dry material is crushed fine, rolls, dry mullers, or Symons disk crushers are used to reduce the material to final treatment size which is about  $\frac{1}{16}$  or  $\frac{3}{32}$  inch. For fine crushing in plants in which an oil-froth flotation method of concentration is used, coarse crushing is followed by some type of rotating mill, ball mill, pebble mill, or rod mill. These machines are sometimes preceded by a Symons disk crusher.

#### CONCENTRATION METHODS.

There are four methods of concentrating graphite ores, water "skin-flotation," the pneumatic process, the log-washer process, and oil-froth flotation.

The extent to which these various processes are used in Alabama is as follows, the figures including plants that contemplate putting in those systems under which they have been classified: The skin-flotation washer system is used in twelve and one-half plants. One of the plants consists of two units, one of which is the Minerals Separation oil-froth flotation. The pneumatic system is used in three plants, the log-washer system in two plants, and oil-froth flotation systems in twenty-one and one-half plants.

## WATER "SKIN FLOTATION."

In the water "skin flotation" system, a typical flow sheet of which is shown in figure 1, the ore is crushed dry to  $\frac{1}{16}$  or  $\frac{3}{32}$  inch. It is

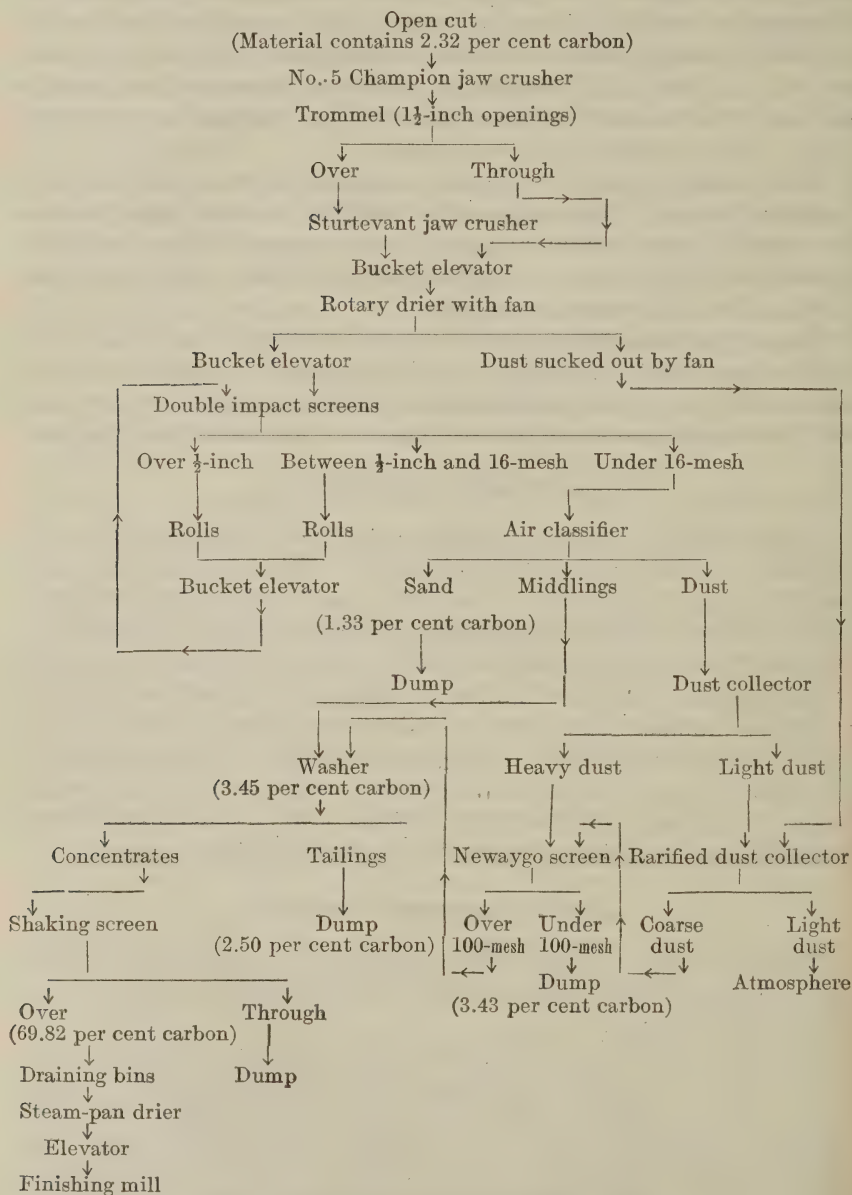


FIGURE 1.—Flow sheet of water "skin flotation" system.

then sent through a rotary drier, screened, and passed over an air separator where it is classified into three products—"scalpings" or

tailings, that go to the waste dump; middlings, that form the "washer" feed; and fines, that are generally treated on a separate group of "washers." The middlings plus the fines, which form 50 to 65 per cent of the crude ore, go to a bin for fine ore, and are fed to "surface-tension" washers where the graphite and some of the mica are floated off, on the surface film of water. The gangue minerals are wet by the water, sink to the hutch of the washer, and are discharged to the tailings launder. The rough graphite concentrate is then dewatered on a trommel or shaking screen, which also removes both the graphite too fine to make No. 1 flake and the fine sand that has been carried over into the concentrates by the film formed by the graphite on the surface of the water. The dewatered concentrate, a dry sample of which analyzes 40 to 60 per cent graphitic carbon, is then dried in a rotary or steam-pan drier and sent to the finishing plant. Average figures for the amount of rough concentrate recovered per ton of ore are not available, but depend of course in large measure upon the grade of ore and of concentrate.

The types of screening and air-classifying apparatus used differ widely, and many of them are unique. Newaygo and "whip-tap" screens are most popular, but other types of impact and shaking screens are used, as well as hexagonal trommels. Most of the air-classifying apparatus is individually designed and erected, and gives results that seem to be efficient. Practically nothing has been done to check up the work of this class of machine; its operation is judged entirely by looking at the products without having even the "scalpings" analyzed. So far as is known, no sufficient quantity of No. 1 flake would be recovered to justify additional treatment, with a view to obtaining more No. 1 flake from these "scalpings."

The result of sampling crude ore, or crude concentrates, and "scalpings" is shown in the table following. All samples were analyzed at the Pittsburgh experiment station of the Bureau of Mines.

*Results of operation of air classifier.*

[Figures represent percentages of graphitic carbon.]

Heading.	Graphitic carbon, per cent.	Graphitic carbon in "scalpings" or tailings, per cent.
Operating on crude ore:		
First sample.....	3.45	1.35
Second sample.....	2.55	1.01
Operating on crude concentrates:		
First sample.....	62.80	4.68
Second sample.....	48.80	1.83

There are three types of "skin flotation" washers sufficiently distinct in principle to permit individual description. Figures 2, 3, and 4 illustrate the underlying principles in design. The dimensions given are estimates and not actual measurements.

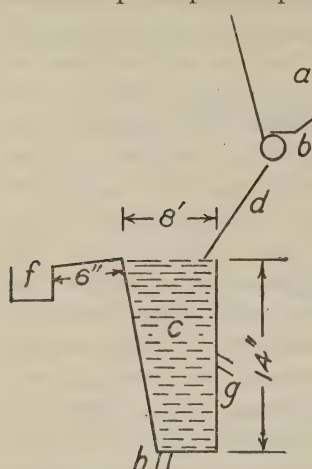


FIGURE 2.—Section through original Munro washer (dimensions indicated are approximate). *a*, Stock bin; *b*, feed cylinder (2-inch revolving shaft); *c*, lake 8 inches wide; *d*, baffle inclined 50° to 65° from horizontal, 10 inches long; *f*, collecting launder, into which graphite is carried over an apron from *c*; *g*, water inlet; *h*, hutch spigot.

#### MUNRO WASHER.

The Munro washer is a rectangular washer in which the feed drops from the feed cylinder to an inclined flat baffle and thence to the water film that carries the graphite to the collecting launder and permits the gangue to sink to the hutch. These washers are built in units about 4 feet long, placed end to end and back to back.

#### COLMER WASHER.

The Colmer washer is a circular washer with a conical feed baffle. The ore falls from the baffle on a revolving circular disk with ribs cast on it. The tangential motion thus imparted to the ore is intended to assist in carrying off the graphite by reducing the tendency of the falling particles to rupture the surface film of water. Some operators claim that better results are obtained with this washer by keeping the disk stationary, relying on the disk solely to change the direction of travel of the particles of ore. These operators claim that the tangential motion given by the revolving disk produced a lower-grade concentrate than was obtained by keep-

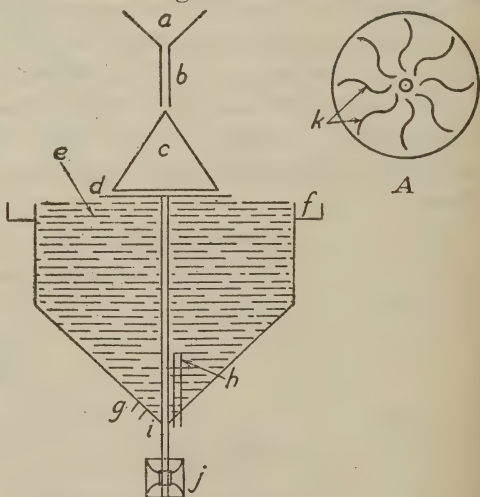


FIGURE 3.—Section through Colmer washer (not drawn to scale). *a*, Ore bin; *b*, feed pipe; *c*, conical feed plate; *d*, revolving disk; *e*, lake about 9 inches wide; *f*, collecting launder; *g*, hutch spigot; *h*, water inlet; *i*, shaft; *j*, pulley for rotating disk; *k*, ribs. A, detail of disk *d* (about 18 inches in diameter).



ing the disk stationary. As shown in figure 3, the washer is about 42 inches in diameter.

#### NEW MUNRO WASHER.

The new Munro washer (fig. 4), which also is circular in shape, is 6 to 12 inches larger in diameter than the Colmer, most of this increase in size being taken up by increasing the width of the "lake." In the Munro washer, the treatment water is forced through an annular opening, formed by placing a cast-iron cover over a bell-mouth casting which is attached to the feed-water pipe. This washer does away with the mechanical motion of the Colmer type. It is difficult, however, to keep an unobstructed flow of water through the annular opening.

#### PNEUMATIC PROCESS.

In the treatment of graphite ores by the pneumatic process, pneumatic apparatus is used throughout. A typical flow sheet for this system is not given, because the system has been successful in only one mill. The details of the operation of this plant can not be made public.

#### LOG-WASHER PROCESS.

A typical flow sheet of the log-washer process is shown in figure 5. In this method of treatment the crude ore is crushed and screened without drying. It is then treated in log washers, kerosene oil being added to the water in the apparatus. The concentrates are screened, washed on a cement floor, drained, and dried. This system eliminates the expensive preliminary drying of the crude ore, necessary with the water "skin flotation."

The simplicity and effectiveness of this plant are clear to the observer. The grade of rough concentrate made is excellent and the only important loss occurs in the coarse-sand tailings, which with

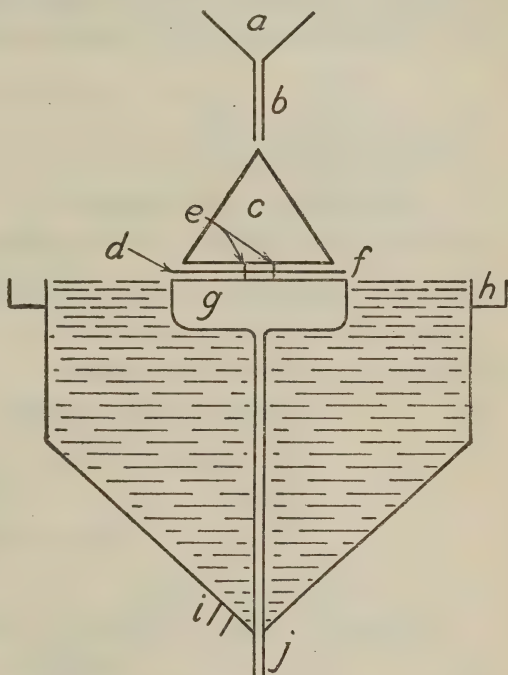


FIGURE 4.—Section of new Munro washer (not drawn to scale; lake about 12 inches wide). *a*, Stock bin; *b*, feed pipe; *c*, conical feed plate; *d*, cover plate; *e*, set screws for regulating width of opening; *f*, annular opening; *g*, bell mouth; *h*, collecting launder; *i*, hutch spigot; *j*, feed-water pipe.

regrinding might be made to yield more flake graphite. A test of the coarse-sand tailings, in the same way that has been suggested under the discussion with reference to the "scalpings" of air classifiers, would determine the desirability of regrinding. The fine-sand tailings are remarkably free from detached flakes of graphite.

#### OIL-FROTH FLOTATION SYSTEMS.

The majority of the graphite plants in Alabama have installed some type of oil-froth flotation cell. Eight plants have installed the Callow pneumatic cells, four have installed the Simplex type of cell, three and one-half (a two-unit plant has one skin-flotation

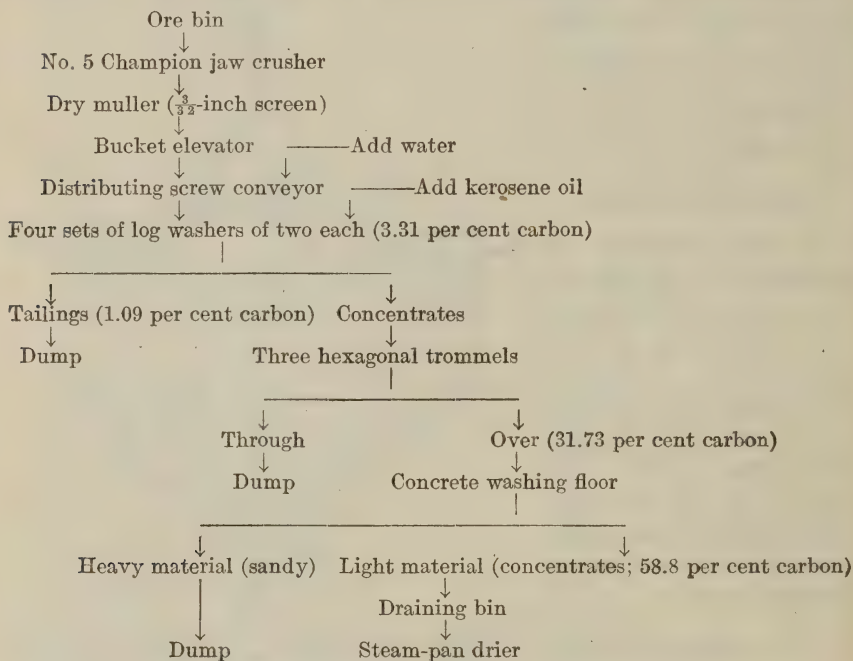


FIGURE 5.—Flow sheet of log-washer process.

washer system) have installed the Minerals Separation cells, and six plants have installed, or intended to install, homemade cells or washers combining some of the principles of these processes.

The chief difficulty with oil-froth flotation systems is in the loss of the large flake. In recovering the large flake, so low grade a concentrate is obtained that an added burden is placed on the finishing plant. There is no difficulty in obtaining a high recovery of the high-grade dust, but there is at present no ready market for this character of material. The best results obtained in the district are from the homemade cells or washers mentioned.

In both the Minerals Separation and the Callow systems, fine crushing is accomplished by Marcy, Lehigh, Marathon, or Hardinge ball or pebble mills. In practically all of the plants in which these two systems have been installed, water classifiers of the Deister or Gemmell type are used. Two plants have Dorr classifiers. For dewatering the rough concentrate, vacuum filters of the Portland or Oliver type are generally used rather than shaking screens.

#### MINERALS SEPARATION SYSTEM.

A typical flow sheet of the Minerals Separation system is shown in figure 6. Assay results for this system are not available.

#### CALLOW PNEUMATIC SYSTEM.

A typical flow sheet of the Callow pneumatic system is shown in figure 7. This system has received greater development in the Alabama district than any other oil-froth flotation system. At one plant the tailings from the Callow cells are treated in an improvised washer with satisfactory recovery.

Samples obtained in Callow plants did not make a good showing, chiefly because of the physical difficulties in taking them. The results of sampling follow:

#### *Results of operation of Callow pneumatic cells.*

[Figures represent percentages of graphitic carbon.]

Test.	Crude ore.	Crude concentrates.	Tailings.
No. 1.....	2.24.....	43.66	Sample "salted."
No. 2.....	Sample lost.....	52.59	No samples possible.
No. 3.....	No sample taken <sup>a</sup> .....	57 to 80	0.5 to 0.9.

<sup>a</sup> At this plant the flake seemingly is being subjected to excessive grinding, with the production of a large quantity of graphite, which will not produce No. 1 flake.

#### SIMPLEX SYSTEM.

The Simplex system includes crushing as well as concentration. The crushing apparatus has no particular advantage over other crushing machinery but the concentration method appears to be efficient.

In this system, the pulp comes unclassified to the center of an elliptical washer on the surface of which jets of water, with flotation oils, are forced at a pressure as high as 40 pounds per square inch. The jet of water entrains air, and a froth is formed which floats the graphite over the side of the washer, while the gangue drops to the hutch.

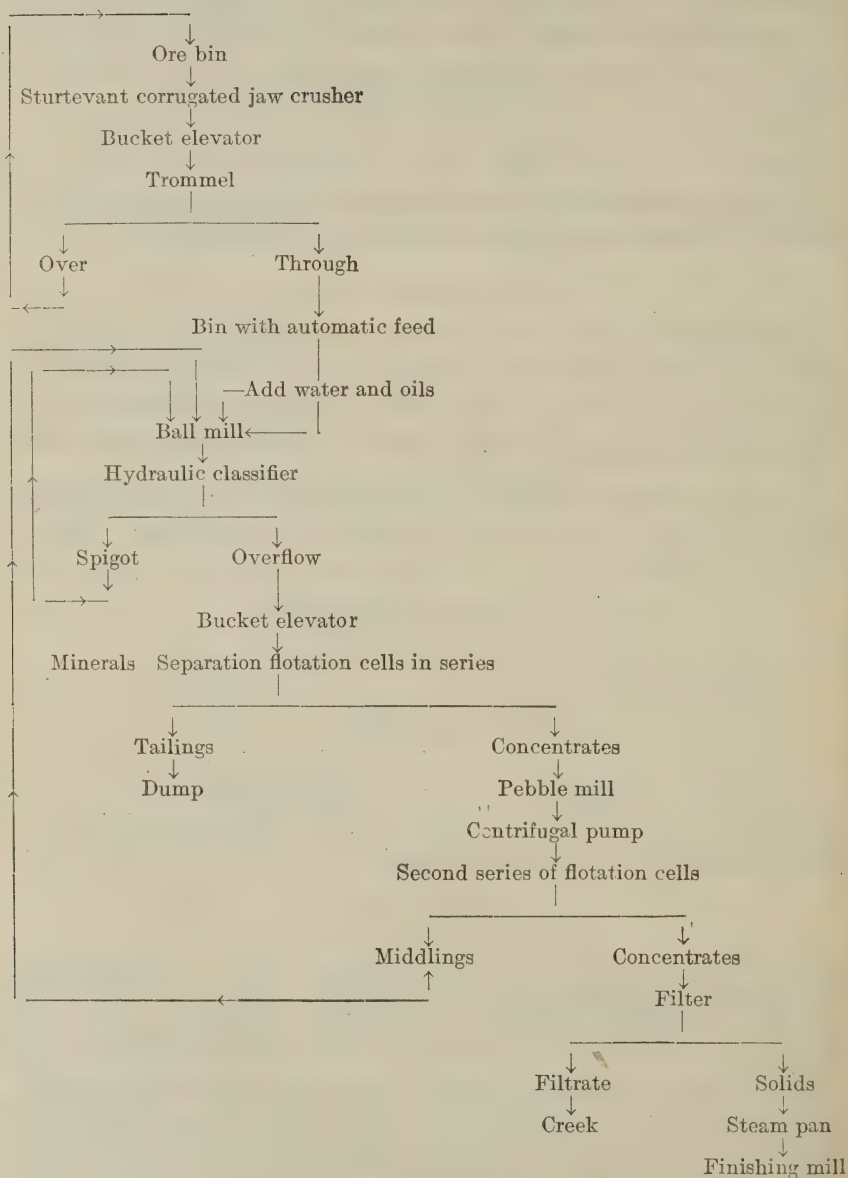


FIGURE 6.—Flow sheet of Minerals Separation system.



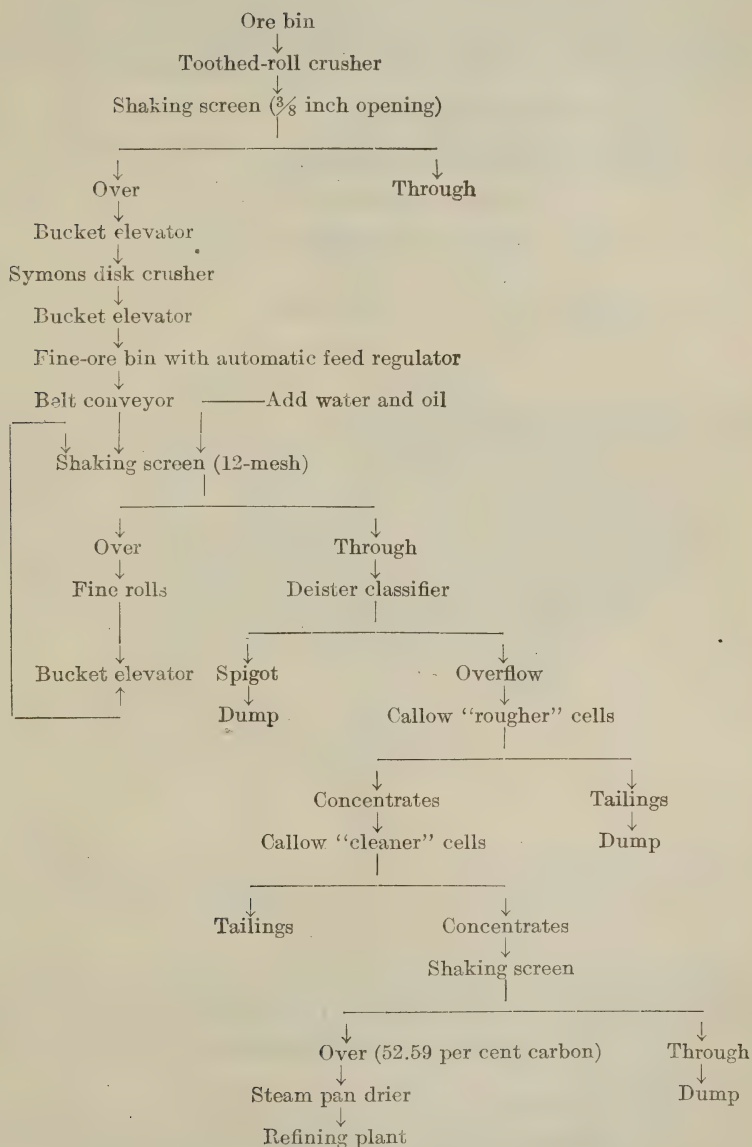


FIGURE 7.—Flow sheet of Callow pneumatic system.

A typical flow sheet for a plant of this type is shown in figure 8, and a sketch of a section through the washer is shown in figure 9.

In various mills in Alabama, Wilfley and Deister tables are being installed to raise the grade of the rough concentrate and to remove gangue from the concentrate fed to the finishing plant. The gangue wears the surfaces of the buhr stones, thus necessitating their more frequent dressing, an operation that is both tedious and expensive.

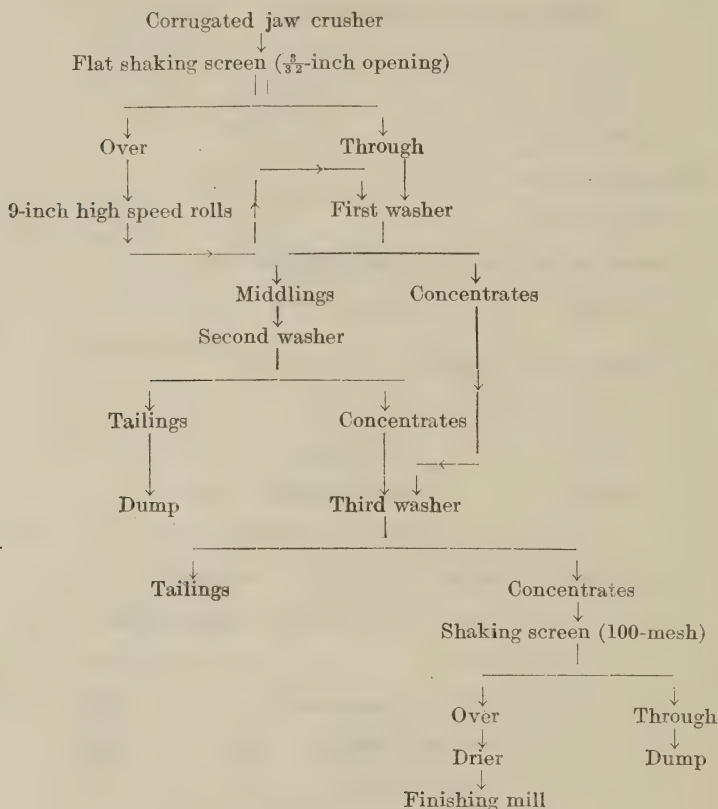


FIGURE 8.—Flow sheet of Simplex system.

At one of the plants, concentrates from a table are treated in an improvised pebble mill. The dust is then removed, and the material is passed through a series of screens, producing No. 1 flake ready for market.

The most important consideration as regards the use of concentrating tables is their capacity. As the use of tables in this district is still in the experimental stage, reliable data as to the success of this new practice is not available.

## REFINING METHODS.

The rough concentrate as it comes from the concentrating plant contains a minimum of 40 per cent graphitic carbon. In some plants, this crude concentrate analyzes as high as 75 per cent graphitic carbon, although the usual proportion is 40 to 60 per cent. The impurities are quartz, mica, wood fiber, etc. Material of this character is not readily marketable except at prices far below those obtained for No. 1 flake; moreover, the market for crude concentrate is limited. These conditions, therefore, necessitate the refining of crude concentrate to No. 1 flake, containing 85 per cent graphitic carbon, and remaining on a No. 8 silk cloth of about 86 mesh.

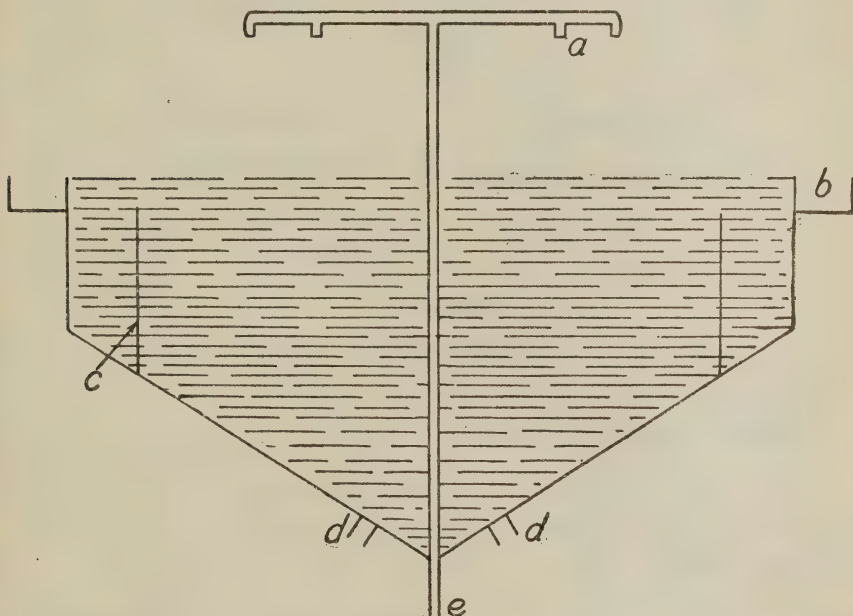


FIGURE 9.—Section of Simplex washer (not drawn to scale; major axis is about 6 feet and minor axis about  $3\frac{1}{2}$  feet). *a*, Spigots,  $\frac{1}{4}$  to  $\frac{3}{8}$  inch orifice, water and oil under pressure; *b*, collecting launder; *c*, screen baffle; *d*, hutch spigots; *e*, feed water.

Microscopic examination shows that the graphite flakes consist of thin laminae of graphite and that present between the laminae are associated minerals, chiefly quartz and mica, which must be eliminated. There are two general methods of refining:

(1) If the crude concentrate comes to the finishing plant with most of the impurities still interlaminated with or attached to the graphite, grinding in a pebble or buhr mill is necessary in order to loosen the graphite from the impurities.

(2) On the other hand, if the crude concentrate comes to the finishing plant with the larger part of impurities as detached par-

ticles, refining can be done pneumatically or with electrostatic machines. Where this method is employed, crushing in the concentrating plant has been of such a character as to accomplish what is usually done in the finishing plant with the buhr or pebble mill.

The first method is the one most generally used; the second method is used in only two plants, and its application is not broad enough to warrant more detailed description at this time.

A flow sheet of a refining plant using the first method is shown in figure 10. This flow sheet is not typical because most plants are

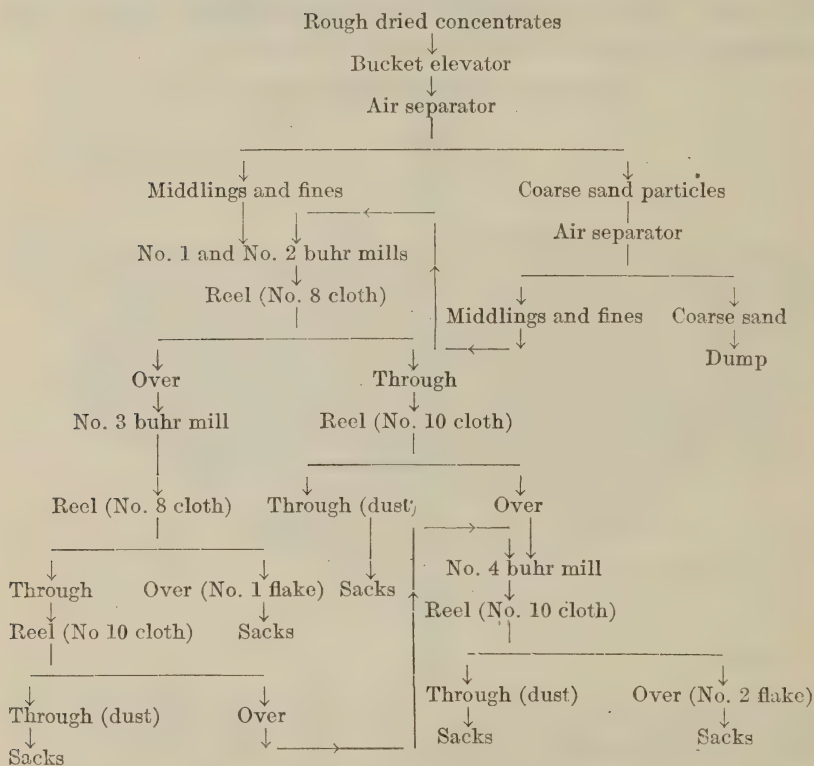


FIGURE 10.—Flow sheet of graphite refining plant.

quite dissimilar, but it represents in a general way the underlying principles governing refining.

#### AIR CLASSIFICATION.

Air classification is used in most of the refining plants and forms an important step in the refining process. Figure 11 shows sketches of air classifiers. Such apparatus is usually improvised. So far as is known no experimental work has been done by the large majority of operators with a view to determining the most efficient proportions and shape for this device.



In principle air suction created by a fan at the back end of each chamber is applied to a falling stream of graphite ore or concentrate. The heavier material, quartz grains with attached graphite, is practically unaffected by the air suction, and falls into the hopper farthest from the fan. The lighter particles are drawn nearest the fan, which sucks the finest dust into the atmosphere. When this

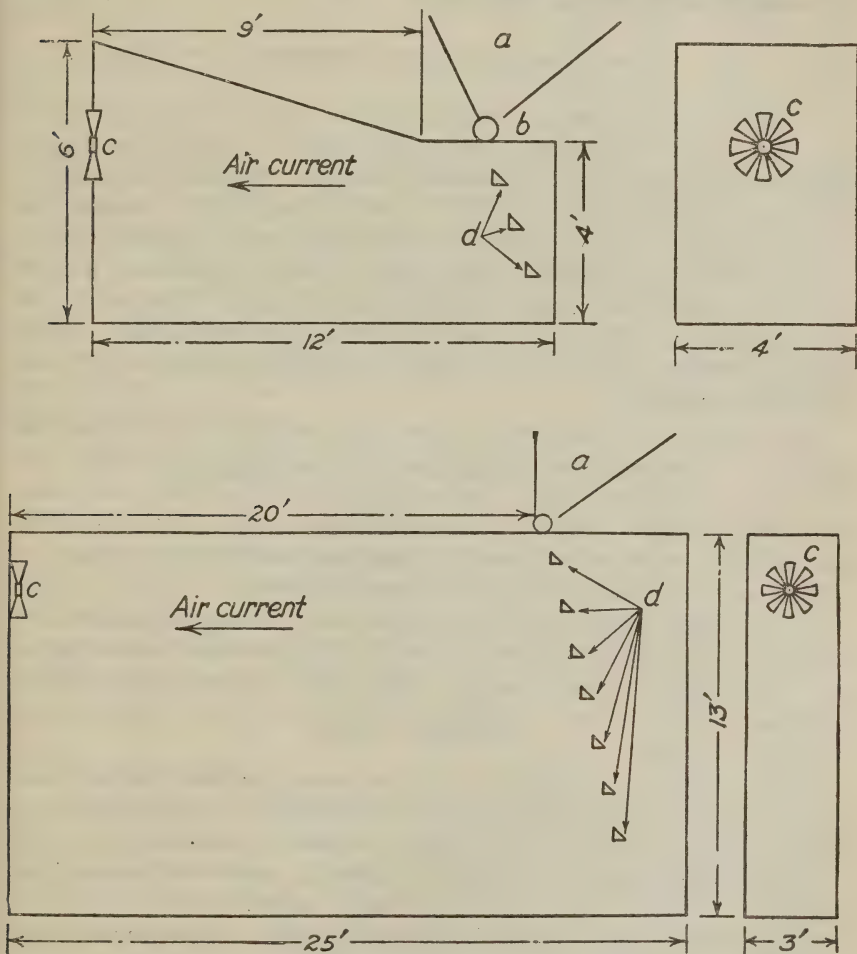


FIGURE 11.—Two types of air classifiers (not drawn to scale). *a*, Stock bin; *b*, feed shaft; *c*, fan; *d*, baffles. Three sets of hoppers are provided for the three products usually made.

apparatus is used on crude concentrates, the middlings—which include the heaviest flake—form No. 1 flake stock, going to the No. 1 buhr mills; the finer material nearest to the fan is screened to remove as much finished No. 1 flake as possible.

In all refining methods the following steps are taken to facilitate the highest possible recovery of No. 1 flake:

1. All dust is eliminated from the circuit as soon as practicable.
2. No. 1 flake is removed from the system as soon as it has been made.
3. As little concentrate as possible is sent to the grinding apparatus.

For analyses of products see discussion of air classification under "Concentration Methods," page 15.

#### RECOVERY OF FINISHED PRODUCTS.

In discussing recovery in graphite plants it should be pointed out that the individual plants are small mills in which systematic sampling and weighing of products are not regularly employed. No data, therefore, are available as to the extraction obtained at various stages of concentrating and refining. It is impossible, also, to state where the greatest losses occur in these operations except from observation and rough sampling in the field. The results of this sampling are indicated wherever practicable on the flow sheets presented herein; the points at which the greatest losses occur have also been indicated.

The only products actually weighed and sampled are the finished products. It would be entirely feasible to sample regularly the crude ore, the tailings from the roughing mill, and the crude concentrates. Various other products could be sampled from time to time, and this work would amply repay the trouble of sampling.

Recoveries of finished materials are as follows:

1. No. 1 flake analyzing 85 to 90 per cent graphitic carbon and remaining on a No. 8 (86-mesh) or a No. 12 (125-mesh) silk cloth—the smaller flake being obtained when a 90 per cent graphite is produced: 10 to 30 pounds is obtained per ton of ore. The average is about 19 pounds.

2. No. 2 flake, analyzing 75 to 80 per cent graphitic carbon and passing through the meshes above mentioned: 2 to 15 pounds is obtained per ton of ore. The average is about 5 pounds.

3. Dust, analyzing 30 per cent or more in graphitic carbon: An average of about 3 pounds per ton of ore is obtained.

If there were a market for No. 2 flake and dust, a greater recovery of these materials per ton of ore could be made.

The recovery percentages for all products are shown in the table following, which represents average milling practice in the district:

*Data showing average recoveries of graphite products in Alabama mills.*

Material.	Carbon.	Weight.	Weight multiplied by per cent.	Recovery.
	<i>Per cent.</i>	<i>Pounds.</i>		<i>Per cent.</i>
No. 1 flake .....	87	19	1,653	33.06
No. 2 flake .....	77	5	385	7.79
Dust .....	30	3	90	1.80
Total finished products .....		27	2,128	42.56
Tails (by difference) .....	1.46	1,973	2,872	57.44
Crude ore .....	2.5	2,000	5,000	100.00

The amount of No. 1 flake produced per man per hour ranges from  $4\frac{1}{2}$  to 8 pounds, and averages about 6 pounds, figured on the total number of men and officials employed. Proportionate quantities of No. 2 flake and dust are obtained. There is no definite advantage apparent from figures obtained from the different classes of plants. The outputs, of course, depend to a large extent on the amount of graphite originally present in the ore, so that although these figures are an index of costs, they can not be used as a measure of comparative efficiency.

#### COMPARATIVE COST OF ERECTING PLANTS.

The cheapest type of plant to install is that using a log washer, and this is probably the cheapest to run. No figures are available as to either initial or operating costs for a plant of this kind.

For a plant capable of treating 10 tons per hour, the initial cost ranges from \$35,000 to \$60,000 erected, not including that of a finishing plant, which would add \$5,000 to \$10,000 more to the cost.

#### FIELD SAMPLING.

In the flow sheets accompanying this report, available assays of samples taken in the field have been inserted, and these give some idea as to the relative efficiency of different methods of milling and of steps in the various processes. In many instances where assays have been omitted, no samples were taken; and sometimes the samples taken were lost either in transit or in drying in the field, or the assay results were inconclusive because of the difficulty of getting a representative sample in the plant.

#### GRAPHITE INDUSTRY IN NEW YORK.

##### PLANTS.

The mining of graphite in New York was started about 60 years ago. This State for many years produced more graphite than any

other, but since 1915 has yielded first place to Alabama, where the growth of the industry has been remarkably rapid.

In 1918 there were three mines in operation, namely, the mine of the Graphite Products Corporation, 3 miles north of Saratoga Springs; Hooper Bros. mine, 4 miles west of Whitehall; and the mine of the American Graphite Co., worked by the Joseph Dixon Crucible Co., 4 miles west of Hague.

#### OCCURRENCE OF ORE.

In New York the ore averages 4 to 6 per cent graphitic carbon as compared with  $2\frac{1}{2}$  per cent in Alabama. There are two important types of graphite deposits in the Adirondack foothills of New York—those on the contact of a limestone and pegmatite, and those on the contact of the Hague garnet-sillimanite gneiss with the Faxon limestone or Swede Pond gneiss. In deposits of the former type, large flakes of graphite are obtained, but such deposits are extremely pockety, and their development has not been profitable. Present production is from the second type of deposit, in which the ore is quite regular and is of a more uniform grade.

The ore on the whole is much harder and more siliceous than the Alabama ore and resembles in many respects the unweathered "blue rock" of Alabama, the original bluish Talladega schist.

#### MINING METHODS.

Graphite ore is mined in New York both in open pits and underground. The larger part of the ore comes from underground mining. All drilling is done with power drills. On account of the siliceous character of the ore, a greater quantity of fines is produced during crushing than is obtained in Alabama. The ore-bearing zone is usually not over 25 feet thick and averages about 15 feet.

#### OPEN-PIT MINING.

Wherever possible, the open-pit system of mining is used because of the ease and cheapness of preparing an open pit, where there is no excessive amount of overburden that requires stripping.

#### UNDERGROUND MINING.

The deposits dip  $25^{\circ}$  to  $35^{\circ}$  so that it soon becomes necessary to go under cover for the ore in the course of the development of any mine. This has its advantage in the North because of the long and severe winters which hamper open-cut work.

The system of underground mining at the largest mine in the district is the room-and-pillar method with underhand stoping. In new developments in the district, a definite system of mining has not as yet been determined.



## CONCENTRATING METHODS.

A flow sheet of one of the plants in New York is given in figure 12. In this plant gravity stamps are used as fine crushers and "buddles" as concentrating machines. Concentrating tables are used for treating "buddle" middlings, whereas, in Alabama they are to be

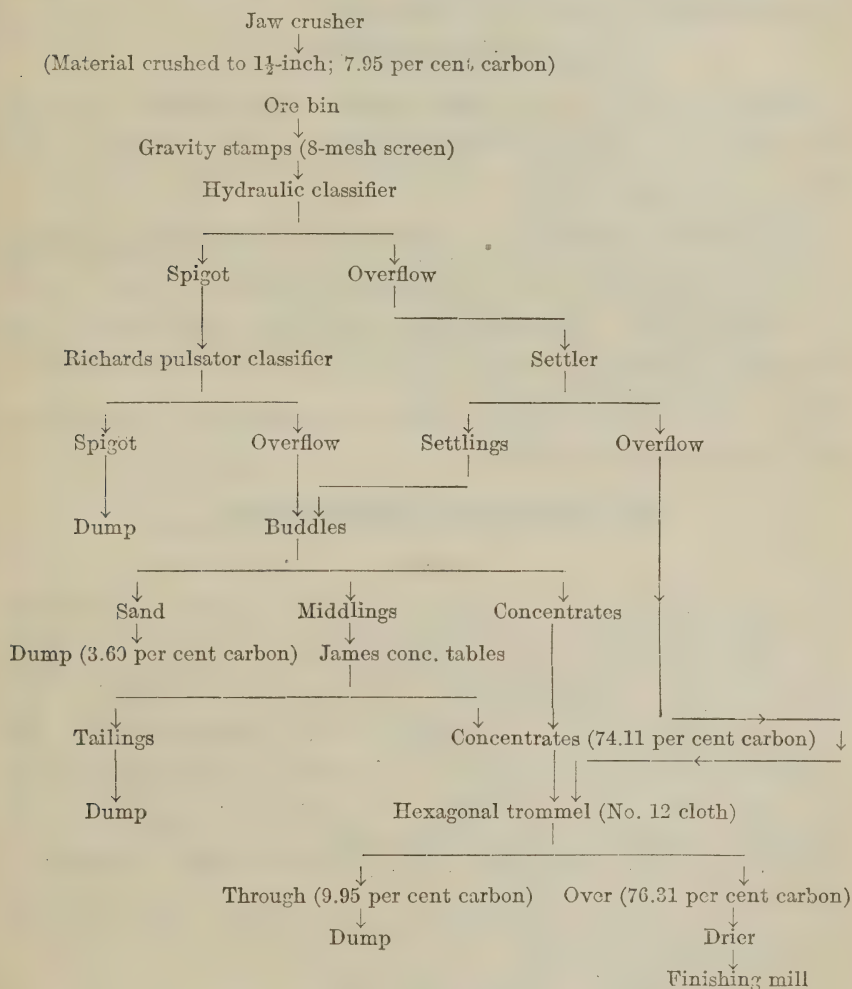


FIGURE 12.—Flow sheet of New York concentrating plant.

employed in raising the grade of rough concentrates preparatory to treatment in the finishing mills.

With crude ore that analyzed 7.95 per cent graphitic carbon, "buddle" tailings analyzed 3.60 per cent C. Tailings from various other machines analyzed 3.96 to 9.95 per cent C., the latter product

consisting of the fines through the rough concentrate trommel. The crude concentrate analyzed 76.31 per cent graphitic carbon.

It should here be noted that the "buddle" is being replaced at the Jos. Dixon Crucible Co. plant by a system of oil flotation. As the product from this mine goes into the manufacture of materials other than crucibles, it is desirable to get as high a recovery of graphite of various grades as possible per ton of ore.

#### REFINING METHODS.

Refining methods do not differ greatly from methods used in Alabama plants. One variation is the use of the Hooper air jig instead of air classification. The chief disadvantage ascribed to this jig is its small capacity.

The American Graphite Co. refinery at Ticonderoga prepares foreign and domestic graphite, both amorphous and crystalline, for use in the manufacture of commercial articles, such as paints, foundry facings, pencils, lubricating flake, and boiler compounds. The by-products obtained from this mill are all utilized in the manufacture of the commodities mentioned.

Data as to recoveries of No. 1 flake, No. 2 flake, and dust in New York practice are not available.

### THE INDUSTRY IN PENNSYLVANIA.

#### PLANTS.

The graphite industry in Pennsylvania has in the past suffered from "wild-cat" and "get-rich-quick" schemes that for a time seriously interfered with the production in the Pickering Valley, Chester county. During the war, this district, in common with other producing districts, responded to the increased demand for graphite. In 1918, five plants were in operation, two close to Byers and three near Chester Springs. The most recent installation is the Huff electrostatic system at the plant of the Rock Crucible Graphite Co., the contemplated flow sheet of which is given in figure 13.

At present (August, 1919) graphite production in Pennsylvania is practically at a standstill.

#### OCCURRENCE OF ORE.

The ore, with one exception, is a soft disintegrated schist dipping from 35° to 50° and analyzing 3½ to 4½ per cent graphitic carbon. In some places most of the gangue has been removed by weathering, the graphite remaining behind as large coarse flakes.

No 4½ Champion jaw crusher (crushing to 2-inch)

24-inch Symons disk crusher

Soft ore added Bucket elevator

Rotary drier (magnesia lined)

Bucket elevator

Rolls 30 by 16 inches (crushing to ½-inch)

Bucket elevator

Newago screen

Through

Over

20-inch by 14-inch rolls

Newago screen (20-mesh)

Through

Over

Bin

Bucket elevator

Air classifier

Dust

Coarse middlings

Fine concentrates

Treatment not worked out

Screens (mesh?)

No. 1 and No. 2 static "roughers"

No. 3 and No. 4 static "roughers"

Through

Over

Waste

Tailings

Concentrates

Concentrates

Tailings

Dump

No. 1 and No. 2 static "cleaners"

No. 3 and No. 4 static "cleaners"

Dump

Concentrates

Middlings

Middlings

Concentrates

Finishing mill

Rolls

Finishing mill

FIGURE 13.—Contemplated flow sheet for Huff electrostatic system.

## MINING METHODS.

With the exception of the operations of the Standard Carbon Co. and part of the operations of the Rock Crucible Graphite Co. all mining is done in open pits. Loading of broken ore is done either by hand or by steam shovel, and little or no drilling is necessary because of the character of the ore.

At the mine of the Standard Carbon Co. the ore is mined in narrow rooms with alternating pillars. The deposit averages 25 feet thick and dips about 40°. The mine is developed by a 120-foot vertical shaft, with levels at 80 feet and 120 feet below the collar. The ore is readily drilled with an auger drill.

## CONCENTRATING METHODS.

In this district three of the concentrating systems described under the outline of the Alabama industry are used. Three plants use the log washer or a similar device; one plant is using the dry process in the form of the Huff electrostatic system, and a fifth plant was using the oil-froth flotation system with the K and K type of cell, but has now discarded this type of cell.

At two of the plants a machine called the "oscillator" is used in place of the log washer. The "oscillator" comprises an inclined table with a series of rakes suspended over it and attached to an eccentric. In addition to the horizontal motion each row of teeth is given a circular motion in a vertical plane, thus stirring the pulp and releasing the graphite, which floats to the lower end of the table while the tailings are worked to the upper end by the rakes. In principle, therefore, this device resembles the log washer. As with the log washer, kerosene oil is used to help float the graphite. Samples taken in plants in which machines of this kind are used gave the following results:

*Results of operation of oscillators.*

[Figures represent percentages of graphitic carbon.]

	Test 1.	Test 2.
Crude ore.....	4.52	3.56
Washer tailings.....	2.19	1.79
Rough concentrates.....	62.00	72.07

The contemplated flow sheet of the plant in which the Huff electrostatic system has been installed is shown in figure 13. The other four plants have flow sheets similar to those described under Alabama practice.

## REFINING METHODS.

Methods of refining used in the Pennsylvania field are similar to those in use in other districts. Two refining plants are being oper-



ated. In the new Huff electrostatic plant it is expected that no additional refining apparatus except dusting and bolting screens will be required. The plants not operating refineries sell their rough concentrate, representing material coarser than 100 mesh and analyzing 60 to 70 per cent graphitic carbon, to an Eastern refiner.

In this district data as to the recovery of finished products are not available. From 50 to 60 pounds of rough concentrate, analyzing as shown in preceding table, are recovered per ton of ore.

## THE INDUSTRY IN TEXAS.

### OCCURRENCE OF ORE.

In Texas graphite is found in the siliceous pre-Cambrian Packsaddle schist in the central region of the State between Llano and Burnet. In many places, the graphitic schist has been disrupted by granite and pegmatite intrusions. This schist, according to Paige,<sup>a</sup> is a metamorphosed sedimentary rock in which the carbonaceous materials have been recrystallized to form graphite.

The ore analyzes 6 to 10 per cent graphitic carbon, but the recovery of No. 1 flake is comparatively low because of the presence of amorphous graphite and very small flake in the ore; moreover, the gangue is hard, so that in crushing, a large amount of the flake is destroyed by being ground to finer sizes than are used as No. 1 flake.

In 1918, three properties were in operation in the State, but in June, 1919, only one was active.

### MINING METHODS.

At the properties of the Southwestern Graphite Co., 9 miles west of Burnet and the Burnet-Texas Graphite Co., 7 miles west of Burnet, open pits have been developed; but at that of the Dixie Graphite Co., 4 miles northeast of Llano, mining is by underground methods.

At the property of the Southwestern Graphite Co., the overburden, which varies in thickness from 2 to 8 feet, is a hardbaked detritus containing large boulders of ore. It contains much clay and vegetable fiber and is said to average 3 or 4 per cent graphitic carbon, only a small part of which can be recovered as No. 1 flake stock. Accordingly, this material is loosened by means of a plow aided by a snatch block and is dumped as waste. In loosening overburden of this character, blasting powder might advantageously be used.

<sup>a</sup> Paige, Sidney, U. S. Geol. Survey, Geol. Atlas, Llano-Burnet folio (No. 183), 1912, p. 4.

When the mine was visited the ore body was being crosscut, preliminary to the development of two working faces along the strike of the ore, which is 80 to 100 feet thick and dips about 65° southeast. The outcrop can be followed on this company's property with a slight interruption for almost three-fourths of a mile along the strike.

In mining, a Keystone wood-fired churn drill is used for drilling holes 5½ inches in diameter. These are placed at the corners of 12-foot squares the first row of which is 12 feet from the last crack of the previous blast. All holes are drilled to a depth of 25 or 30 feet, which is 5 feet below the elevation of the bottom of the open pit. Blasting gelatin of 60 per cent strength is used for breaking. An Erie wood-fired steam shovel with a dipper capacity of five-eighths of a cubic yard loads the broken ore into 5-ton side-dump contractors' cars which are hauled to the mill by a Whitcomb gasoline locomotive. Large boulders are placed to one side by the shovel and are block-holed with self-rotating hammer drills of the jackhammer type.

The Burnet-Texas Co.'s property is 2 miles northeast of that of the Southwestern Graphite Co. on the same schist series and at the time of visit was being worked in a small way. Here the ore is more frequently cut by granite and pegmatite intrusions, and as developed appeared to contain less graphite that would make No. 1 flake stock. Besides an open pit, this property was being prospected by a vertical shaft.

At the Dixie Graphite Co. property, 4 miles northeast of Llano, the ore is won by underground mining. The deposit dips to the southeast at a flatter angle than at the Southwestern Graphite Co.'s property. Near the surface the ore contains considerable quantities of amorphous graphite and fine flake. That mined underground is of better grade as regards the size of flake, and contains considerably more pyrite and mica than were obtained in the surface ore. In the past, much development work was done on this property with a view to mining gold and molybdenum ores, but these efforts were not successful.

#### CONCENTRATING METHODS.

Two plants use the oil-froth flotation method; at a third plant Huff electrostatic machines have been installed. These plants do not differ materially from mills previously described, except that in each a rough concentrate analyzing 70 to 80 per cent graphitic carbon and remaining on an 80 or a 90 mesh screen is recovered.

No refining plants are as yet in operation, although one company is reported to have completed alterations in its mill that will permit production of refined graphite.

In this district the same lack of storage facilities for crude ore was noted as in the Alabama field. The general remarks as to storage also apply to the New York and Pennsylvania districts.

### RECOVERY OF GRAPHITE FROM "KISH."

The product known as "kish" is a mixture of graphite, slag, iron oxide, fragments of iron, and other materials that accumulate about a pig-iron casting ladle, or a Bessemer or open-hearth furnace. The graphite is formed during cooling by the crystallization of excess carbon from molten pig iron. The kish obtained for refining has contained 8 to 10 per cent of graphitic carbon in the form of a light, thin, and fluffy flake.

The method of concentrating and refining kish is similar in many ways to the methods employed with natural graphite. The chief points of difference are that kish is not subjected to crushing, but is carefully screened and is treated with magnetic separators. A patent (No. 1239992) on the recovery of graphite from kish has been issued to F. W. Weisman, and another patent (No. 1271146), to E. C. Ewen.

Kish as a possible source of graphite should not be overlooked. It is questionable, however, whether enough of this material can be obtained to interfere or compete seriously with the mining of natural flake graphite.

### EXPERIMENTAL WORK ON CONCENTRATION AND REFINING.

As is evident from the foregoing discussion, much work had been done on various methods of concentrating graphite ore and refining graphite, but no large amount of effort had been expended on making comparative tests of the processes. It is evident, too, that the possibilities of recovering larger quantities of No. 1 flake have not been exhausted. For these reasons the Bureau of Mines has felt justified in conducting experiments at its Salt Lake City experiment station on the concentration of graphite ore and the refining of graphite. This work is described in the report of F. G. Moses, "Refining Alabama flake graphite for crucible use" (pp. 46 to 74).

### EXPERIMENTAL WORK ON CRUCIBLE MANUFACTURE.

The Bureau of Standards at various times has done some work on crucible manufacture with a view to the use of materials other than the imported clays and graphites. This work has never been carried to a conclusion. It was felt therefore that the Bureau of Mines at its Columbus experiment station should make an investigation that would decide to what extent flake graphite might be used in crucible



manufacture, and would establish physical and chemical specifications for domestic flake graphite.

### PHOTOMICROGRAPHY.

As microscopic examination of crucibles made by various manufacturers, in conjunction with a knowledge of the crucible ingredients, may be of assistance in the solution of the mechanical difficulties at-

tending the use of more than 25 per cent domestic flake graphite in crucible graphite mixtures, this work has been assigned to the Pittsburgh experiment station of the Bureau of Mines.

### METHODS OF SAMPLING.

Generally speaking, the only samples taken in graphite plants are of the finished products. In the past these have not been sampled in the same way by producer and by consumer, so that serious disputes over shipments have arisen, with the result that consumers have often refused to accept certain shipments.

By daily sampling crude ore and crude concentrates, as well as finished products, a much better check on mill work can be obtained than is possible by sampling only the finished grades.

Domestic graphite is usually marketed in sacks holding 150 pounds. The sacks are filled from a spout at the bottom of a small bin to which the products come from the final sizing screen,

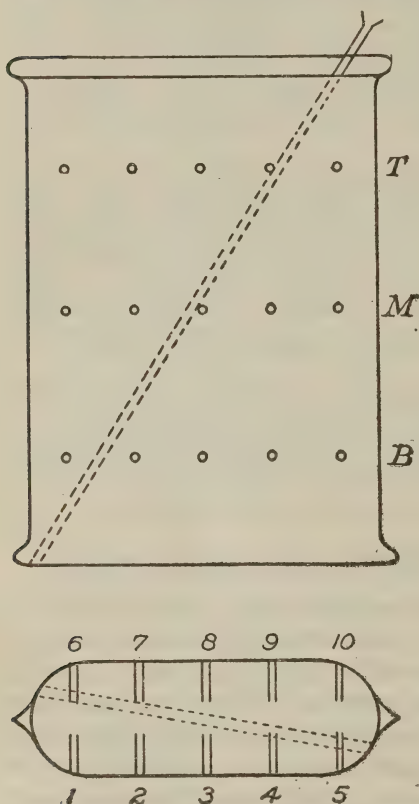


FIGURE 14.—Method recommended for sampling finished sacked product. Correct position of sampling device indicated by dotted lines in two views. The method now generally employed gives samples corresponding to 1 (T, M, B), 3 (T, M, B), or 5 (T, M, B).

which follows the buhr mill. In the process of refining, a product absolutely homogeneous in carbon content is not being made at all times of the day. The material that goes to the finished-product loading bin often varies 5 per cent from the standard determined by the mill operator. The causes of these variations are many. Thus, products obtained just after the buhrstones have been freshly dressed are different from those obtained before the stones have been dressed;



and the feed to the buhrstones varies in character from time to time because of frequent concentrator shutdowns. The stones are adjusted from time to time during the day, but sometimes the character of the No. 1 flake obtained is visibly below the predetermined plant standard. As the sack is filled from a vertical spout, any fluctuations in the grade of graphite in the bin appear as strata in the bag. Moreover the larger flakes naturally fall to the edges of the pile under the spout. The miller usually samples a bag of graphite by taking a spoonful of graphite from the top of the sack or by taking a sample at three points in the bag by means of a small hollow bag sampler about 8 inches long, with an opening 2 inches long near the point.

To show that these methods could not give a representative sample, two tests of graphite in sacks were made as shown in the table following and illustrated in figure 14.

*Results of two tests of sampling sacked graphite.*

[Figures and letters are same as in fig. 14.]

Position of sample.	Carbon.	
	Volatile.	Graphitic.
	<i>Per cent.</i>	<i>Per cent.</i>
T (1-5).....	2.46	86.08
M (1-5).....	1.78	87.32
B (1-5).....	1.68	87.65
T (6-10).....	2.25	84.74
M (6-10).....	2.49	88.82
B (6-10).....	2.10	87.63
1 (T, M, B).....	2.56	88.93
3 (T, M, B).....	2.26	84.90
5 (T, M, B).....	2.67	83.71

In view of the considerations mentioned and the sampling results presented in the table, it is recommended that every bag be sampled continuously from top to bottom in the manner indicated in figure 14, by means of one of the devices described below.

One device consists of a tubular sleeve (fig. 15) 30 inches long and  $\frac{3}{8}$  to  $\frac{1}{2}$  inch in diameter, with a continuous slot extending from the point to within 4 inches of the top. Inserted in the sleeve is a rotating tube with a slot to correspond to the opening in the outer tube. Another suggested sampling device is a single tube with a check valve at the bottom for retaining the sample while the tube is being removed from the sack. It is further recommended that all samples be taken obliquely from top to bottom to get a fair proportion of the larger flakes which naturally slide farthest during the filling of the sack, and tend to collect at the sides.

The method of sampling shipments adopted by the Graphite Producers Association of Ashland, Ala., is an honest effort on the

part of Alabama operators to avoid disagreeable and costly disputes with the consumer.

As soon as a quantity of graphite is ready for shipment, the local chemist at Ashland divides the shipment into lots of 25 sacks. Samples are drawn from each bag and then combined to represent the individual lots. Analyses are made of the lot samples, and if any lot falls seriously below shipping grade, it is discarded from the shipment. The grade of the shipment is then calculated by averaging the analyses of the lot samples.

#### METHODS OF ANALYSIS.

The method of analyzing graphite products or ores has never been standardized to the extent of defining volatile and fixed carbon in a manner acceptable to all analysts. Accurate methods used in determining carbon in graphite products are tedious. Work recently

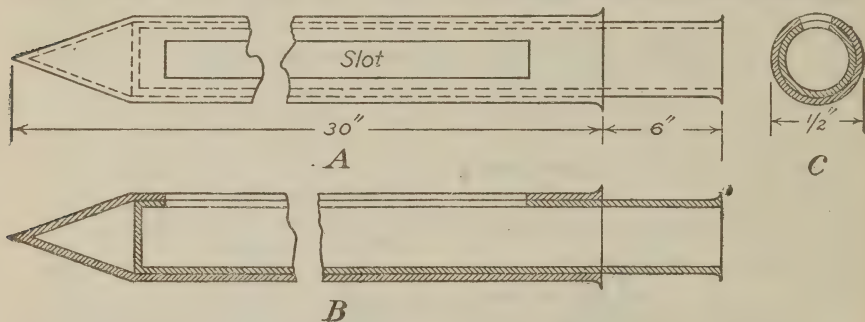


FIGURE 15.—Sections of sampling device (not drawn to scale). A, Longitudinal section, showing slot; B, longitudinal section, samples turned 90° from position A; C, cross section through slot.

done by the Pittsburgh experiment station of the Bureau of Mines on the chemical analysis of graphite products had in view the devising of a rapid, accurate, and simple method of analysis with a fair degree of accuracy. A description of this method is given on pages 43 to 45.

#### TENTATIVE SPECIFICATIONS.

In the past, crucible makers who used domestic flake graphite have as a rule bought this material only after examining samples submitted by producers. It may be helpful to suggest tentative specifications for No. 1 flake graphite.

The table following gives the screen and chemical analyses of graphite that is being used in the experimental work of the Columbus experiment station of the Bureau of Mines on graphite crucibles. All screen analyses were made at the Pittsburgh station of the bureau. A Tyler standard Ro-tap apparatus was used, and the graphite sample was screened for 45 minutes. All chemical analyses

were made at the Pittsburgh station, the volatile-carbon determinations of samples being made by heating for three minutes at 800° C.

*Results of analyses of domestic flake graphite.*

Sample No.	Kind of sample.	Carbon.		Cumulative percentages.				
		Volatile.	Gra-phitic.	Over 20 mesh.	Over 35 mesh.	Over 65 mesh.	Over 100 mesh.	Under 100 mesh.
		<i>Per cent.</i>	<i>Per cent.</i>					
1.....	Artificial graphite.....	1.27	87.82	0.3	11.7	54.9	95.9	4.1
2.....	No. 1 flake.....	1.68	82.02	.....	.1	42.0	94.8	5.2
3.....	do.....	1.32	92.09	.....	2.5	28.0	60.0	40.0
4.....	No. 1 flake and No. 2 flake.....	2.07	86.85	.....	3.0	31.0	74.3	25.7
5.....	No. 1 flake.....	3.65	74.67	.....	3.3	30.3	89.6	10.4
6.....	do.....	.....	.....	.1	15.4	69.6	95.6	4.4
7.....	Dust.....	6.06	41.33	.....	.1	1.2	3.2	96.8
8.....	do.....	7.37	32.41	.....	.....	.....	.2	99.8
9.....	do.....	7.20	31.47	.....	.....	.4	3.9	96.1
10.....	No. 2 flake.....	2.96	80.44	.....	.....	.....	10.5	89.5
11.....	do.....	2.31	81.43	.....	.2	9.9	67.6	32.4
12.....	No. 1 flake.....	.....	.....	.....	5.7	58.5	93.0	7.0
13.....	do.....	1.98	86.18	.1	6.7	56.4	97.0	3.0
14.....	No. 2 flake.....	2.17	86.30	.....	.1	.3	22.2	77.8
15.....	No. 1 flake.....	.....	.....	.....	9.7	62.2	92.4	7.6
16.....	No. 1 flake, special.....	1.08	92.43	.....	8.2	69.6	99.7	.3

Nine crucible manufacturers sent to the Bureau of Mines samples of graphite as prepared by them for incorporation in their crucible mixtures. Screen analyses of these samples were made at the Washington laboratory; the results showed considerable variation. The laboratory lacked the equipment usually found in establishments for the making of screen analyses, and consequently a nest of sieves of 20, 40, 60, 80, and 100 mesh had to be used rather than the standard meshing. For comparison, however, the results will answer just as well.

*Results of analyses of manufacturer's samples at Washington laboratory.*

No. of sample.	Manufacturer.	Cumulative percentages.						Vol-ume of 100 grams (c. c.).	Remarks.
		20 mesh.	40 mesh.	60 mesh.	80 mesh.	100 mesh.	100 mesh.		
1	A.....	1	68½	83½	92½	94½	100½	106	Contains 35 per cent flake and 65 per cent Ceylon.
2	B.....	13½	57½	71½	80½	85½	99½	76	100 per cent Ceylon.
3	C.....	12½	88½	96½	98½	99	100	149	Madagascar flake.
4	C.....	16½	71	81½	89½	90½	100	86	100 per cent Ceylon.
5	D.....	13½	62½	74½	88	93½	99½	81	Contains 10 per cent flake and 90 per cent Ceylon.
6	E.....	23½	66	76	83½	87	100	81	100 per cent Ceylon for steel "pot."
7	F.....	1½	37½	62½	88½	96½	100	145	100 per cent Canadian.
8	G.....	14½	58½	75	85½	89½	99½	95	Contains 25 per cent flake; 75 per cent Ceylon for steel "pot."
9	H.....	4½	49½	69	80½	84½	99½	93	Contains 15 per cent flake and 85 per cent Ceylon.
10	I.....	40	74	84	89½	92	99½	89	100 per cent Ceylon.
11	I.....	16½	59½	75½	86½	90	99½	116	Contains 17 per cent flake and 83 per cent Ceylon.



## CARBON CONTENT.

It is recommended that the graphitic carbon content of No. 1 flake be not less than 85 per cent. By graphitic carbon is tentatively meant the carbon remaining after the dried sample has been burned for three minutes at 800° C.

The table following gives the results of complete analysis of seven samples of crucible graphite. The analyses were made at the Pittsburgh station of the Bureau of Mines. The volatile carbon was determined by heating the samples at 800° C. for three minutes.

*Results of complete analysis of seven samples of graphite.*

Source of graphite.....	Alabama.	Alabama.	Alabama.	Alabama.	New York.	Pennsylvania.	Ceylon.
Sample No.....	1	2	3	4	5	6	7
Volatile carbon.....	2.08	1.40	2.66	1.31	1.30	1.53	1.68
Graphitic carbon.....	84.52	90.58	81.82	91.18	88.97	88.80	85.06
SiO <sub>2</sub> .....	7.02	3.99	8.33	4.08	4.34	5.24	7.81
Al <sub>2</sub> O <sub>3</sub> .....	5.06	2.96	6.35	2.31	2.40	2.05	2.82
Fe <sub>2</sub> O <sub>3</sub> .....	.53	.18	.39	.40	1.08	1.75	1.61
TiO <sub>2</sub> .....	.14	.19	.13	.13	.38	.05	.13
CaO.....					.07		.19
MgO.....	.23	.16	.06	.08	.76	.09	.21
K <sub>2</sub> O.....	.15	.23	.14	.31	.55	.08	.25
Na <sub>2</sub> O.....	.008	.01	.11	.03	.12	.12	.11
SO <sub>3</sub> .....	.007	.14		.01		.21	.005
P <sub>2</sub> O <sub>5</sub> .....	.06			.02	.02	.05	.05
MnO.....						.07	.04
CuO <sup>a</sup> .....		.14					
ZnO <sup>a</sup> .....							.03
Total.....	99.805	99.98	99.99	99.86	99.99	100.04	99.995

<sup>a</sup> Contamination probably from brass screens.

The results presented indicate that the amount of deleterious impurities in domestic flake graphite is small. In the samples of Alabama graphite the Fe<sub>2</sub>O<sub>3</sub> content did not exceed 0.53 per cent, and the combined content of alkalis and alkaline earths did not exceed 0.5 per cent. The amounts of mica and pyrite present were small indeed, judged from the alkali and SO<sub>3</sub> content. Even the Pennsylvania sample with a high Fe<sub>2</sub>O<sub>3</sub> content showed only a small amount of SO<sub>3</sub>, also a low alkali content. The sample from New York showed a combined alkali content of 1.5 per cent, and the Fe<sub>2</sub>O<sub>3</sub> content was slightly more than 1 per cent.

If the single sample of Ceylon graphite analyzed may be taken as representative—and it should be stated that this sample was graphite prepared for use in crucibles—it is evident that, chemically, domestic graphite compares favorably with the Ceylon graphite.

## SCREEN ANALYSIS.

The following tentative specifications as to screen analysis in cumulative percentages are recommended:



*Recommended specifications for screen analysis.*

[Figures represent cumulative percentages.]

Grade.....	No. 1 flake.
Over 35 standard mesh.....	35
Over 65 standard mesh.....	50
Over 100 standard mesh.....	100

Permissible allowance of not more than 3 per cent through 100 mesh.

**PRESENT STATUS OF GRAPHITE MINING.**

The request of the War Industries Board on August 10, 1918, that not less than 20 per cent of domestic flake graphite be incorporated into crucible-graphite mixtures during the balance of 1918, and 25 per cent thereafter, established a market for domestic flake which, it was hoped, would continue after the war. If domestic graphite can be used in this proportion without adversely affecting the quality of crucibles, it will be used provided it can be produced at costs that compare favorably with the prices paid for imported graphites. Under normal conditions the efficiency of crucibles is expressed as so many heats per crucible. However, during the war the vital comparison, as regards the national welfare, was the number of pounds of imported graphite used per heat. The introduction of every pound of domestic flake graphite that did not increase the consumption of imported graphite per crucible heat saved necessary ship tonnage. The proportion of domestic flake that reduces the consumption of imported graphite to a minimum has not yet been determined.

In all of the graphite mining districts in this country operating costs are very high, and in normal times, if imports were unrestricted, the domestic mining industry could not compete with the industries in Madagascar and Ceylon. On both of these islands, the deposits are of large extent and of such high grade that a simple cobbing or washing represents the only milling necessary. Labor is much cheaper in the Far East than in the United States. Therefore, in spite of the great distance from the market, graphite from these islands could be placed on the docks in this country at prices which at present would provide no profit for domestic producers. Also, overproduction abroad might tend to force prices still lower through dumping of surplus production on the American market.

Operating costs in the United States in October, 1918, ranged from 6 to 14 cents, with an average of 10 cents, per pound of No. 1 flake. No allowance has been made for depletion and depreciation, which would add 1 to 2 cents per pound to the costs stated. In arriving at these costs the production of No. 1 flake has been charged with all operating expenses and has been credited with the miscellaneous income derived from the sale of No. 2 flake and dust. The wide

variation in costs is due to differences in the efficiency of individual plants as well as to fluctuations in the grade of ore treated by the different plants.

The graphite in the Alabama district is of lower grade than that in any of the other districts mentioned in this report. With an average recovery of 19 pounds of No. 1 flake per ton of ore and an after-war price of 9 cents per pound, which is 3 to 4 cents higher than the pre-war price, it is evident that, even though all the ore is mined in open pits, only those plants efficiently managed and working comparatively rich deposits will survive unless new and important uses develop for the by-product grades so as to warrant the establishment of a local central factory for producing commercial graphite products. Many of the Alabama plants are 6 to 9 miles from the railroad and are in localities where most of the roads are difficult to maintain because they are of clay and poorly drained. At plants so situated the cost of transportation becomes a serious factor in operating expenses.

Until 1913, the graphite production in Alabama was small, but has steadily increased so that since 1915 Alabama has been the leading graphite-producing State in the Union.

Milling capacity in the Alabama district, with the 39 plants now in operation or building, is sufficient to produce a large quantity of graphite, if these mills could all run 20 hours a day. On the assumption that each mill would average 3,000 pounds of No. 1 flake daily, which is a conservative estimate, a total production of nearly 60 tons a day could be obtained. The actual production of No. 1 flake from this district during 1918 averaged less than 350 tons per month.

In New York and intermittently in Pennsylvania, graphite was mined before the war. New York will undoubtedly continue to produce graphite after the war in somewhat greater quantities than before. Both of these districts are close to a large market. The recovery of graphite per ton of ore is double that obtained in Alabama; this is a distinct advantage, particularly when by-product grades can be sold advantageously.

The costs of production are practically the same as those in Alabama, chiefly because most of the plants in New York and Pennsylvania have just started producing; but with reversion to peace times, these districts have a distinct advantage over the Southeastern district, in spite of the necessity in New York for underground mining.

In the latter part of 1918 the New York district was producing not more than 90 tons of graphite per month, of which probably 60 per cent would be No. 1 flake. The Pennsylvania field produced about 50 tons of graphite per month, of which about 60 per cent would be No. 1 flake.

The Texas district, if provided with an outlet for by-product grades of flake graphite, would have a good opportunity of surviving post-war readjustments. The production from this district has not been large, but could be considerably increased without great difficulty. With respect to a Western and Central market, the Texas deposits are favorably situated. The deposits are not only of large extent but are high grade as well.

These statements have been made in order to indicate, in a general way, the precarious position of the mining of domestic crystalline flake graphite. It would be presumptuous to suggest methods by means of which the mining of domestic graphite can be established firmly on a postwar basis, for underlying the entire consideration is the fundamental economic law of supply and demand. As regards domestic flake graphite, the supply exists and can be obtained. If the demand can be created or extended by scientific, rather than by artificial means, the supply will be forthcoming whenever the price is high enough to justify production.

With the resumption of peacetime activities, the demand for crucibles and other graphite products has slackened. This, of course, has reacted upon the domestic graphite mining industry, with the result that production has been greatly curtailed. Until business in the United States has resumed its normal peace-time trend, some method of control or regulation will be necessary to prevent the collapse of certain industries, which the lessons of the present war have taught should not be permitted to die. To be independent of foreign graphite, the use of 100 per cent domestic flake in crucibles will have to be developed, if the graphite crucible continues to be used as a medium for melting alloys of all kinds. It is possible, of course, that the use of some refractory other than graphite, or the extended use of melting furnaces requiring no crucibles may develop to such a point as to make present practice obsolete.

#### **METHOD FOR RAPID ANALYSIS OF GRAPHITE USED BY THE BUREAU OF MINES.**

By G. B. TAYLOR and W. A. SELVIG.

In the following paragraphs are outlined certain methods of graphite analysis used by the Pittsburgh laboratory of the bureau. These are intended to provide a simple and rapid procedure for the proximate analysis of graphite, such as can be performed in the average laboratory in the minimum of time, together with a fair degree of accuracy. The proximate analysis consists of the determination of moisture, volatile matter, ash, and graphitic carbon by difference.



## PREPARATION OF SAMPLE.

On account of the difficulty of grinding flake graphite, no degree of fineness can be fixed; however, for material containing considerable gangue, grinding to pass through a 60-mesh sieve is recommended. The flakes of graphite separated during the screening must of course be included in the 60-mesh material, and must be thoroughly mixed before portions for analysis are weighed out.

## DETERMINATION.

## MOISTURE.

One gram of the sample is placed in a weighed porcelain capsule  $\frac{7}{8}$ -inch deep by  $1\frac{3}{4}$  inches wide, and heated for one hour at  $105^{\circ}$  C. in a constant-temperature oven. The capsule is then removed from the oven, covered, and cooled in a desiccator over sulphuric acid. The loss in weight multiplied by 100 is recorded as the percentage of moisture. If platinum crucibles are available, it is preferable to use a 25-c.c. platinum crucible instead of the porcelain capsule.

## VOLATILE MATTER.

The covered capsule containing the graphite from the moisture determination is placed in a muffle furnace and heated at  $800^{\circ}$  C. for three minutes, removed from the furnace, cooled in a desiccator, and weighed. The loss in weight multiplied by 100 is recorded as the percentage of volatile matter.

A lower temperature and a shorter time interval than is used in the standard method for the determination of volatile matter in coal is recommended, as the volatile-matter content in graphite is small, and the oxidation and loss of graphitic carbon should be cut down to a minimum.

## ASH.

Ash is determined in the residue from the volatile-matter determination. The capsule containing the residue from that determination is placed in a muffle furnace at about  $800^{\circ}$  C. and is heated until completely burned. The capsule with its contents is removed from the muffle and cooled and weighed, after which it is replaced in the muffle, heated for half an hour, cooled in a desiccator, and weighed again. This heating and weighing should be continued until the change in weights between two consecutive ignitions is 0.0005 gram or less.

The weight of the crucible plus the ash, minus the weight of the empty crucible, multiplied by 100 is recorded as the percentage of ash.



## GRAPHITIC CARBON.

Graphitic carbon, which is determined by calculation, is the difference obtained by subtracting the sum of the percentages of moisture, ash, and volatile matter from 100.

## GRAPHITIC CARBON BY COMBUSTION.

The most accurate method of determining graphitic carbon is by combustion, and subsequent weighing of the  $\text{CO}_2$  formed. While it has been found that graphitic carbon, as calculated, checks very closely with the graphitic carbon determined by direct combustion, the latter should be used when greater accuracy is desired. This method, however, calls for special equipment and requires considerable skill in manipulation.

On account of the possible presence of carbonates in ore, and possibly small amounts of flotation oils in the sample, if a concentrate, a preliminary treatment before combustion is required.

## PRELIMINARY TREATMENT OF ORES.

A sample, weighing 0.2000 to 1.0000 gram, depending on the relative amount of graphitic carbon, is placed in a 100-c.c. evaporating dish with 25 c.c. of 1 to 1  $\text{HCl}$ , heated over a hot plate for 15 minutes, filtered through a filter of ignited asbestos, and washed with hot water until free from chlorides. The filter and the residue are then transferred to a porcelain or platinum boat and dried on the hot plate. The boat containing the residue is transferred to the tube of a combustion furnace and burned in a stream of oxygen gas, the  $\text{CO}_2$  formed being collected in a potash bulb with a 30 per cent  $\text{KOH}$  solution. The combustion tube should contain some fused lead chromate to retain any sulphur that may be present.

## PRELIMINARY TREATMENT OF CONCENTRATES.

Concentrates are likely to contain flotation oils, which must, of course, be removed before combustion, or too high results will be obtained for graphitic carbon.

This may be done by placing a sample weighing 0.2000 to 0.5000 gram in a small Erlenmeyer flask and adding about 25 c. c. of ether, corking loosely, and allowing to stand for about one-half hour, shaking at intervals. The mixture is filtered onto asbestos in a Gooch crucible, the asbestos having previously been washed with hydrochloric acid and ignited. The residue on the filter is washed with alcohol and next with distilled water; it is then ready to be treated with acid for the removal of possible carbonates, as is done in the treatment of ores.

# REFINING ALABAMA FLAKE GRAPHITE FOR CRUCIBLE USE.

---

By FREDERICK G. MOSES.

---

## INTRODUCTION.

In order to determine the possibility of producing a satisfactory crucible graphite from Alabama flake graphite, an investigation embracing a large number of tests was made at the Salt Lake City station of the Bureau of Mines. The following report contains the information obtained during the progress of this investigation.

A primary object of the work was to ascertain the most satisfactory means by which the low-grade graphite concentrates produced from Alabama ores could be made into satisfactory crucible stock. Of course, the success of any one of the methods of treatment developed will depend both on the amount of satisfactory material that can be recovered by the finishing process, and on the ease and cheapness of the operation itself. In all of the experiments both of these factors were kept in mind.

## REQUIREMENTS FOR CRUCIBLE GRAPHITE.

After an extended investigation into the specifications for graphite satisfactory for use in crucible manufacture, Geo. D. Dub, has suggested the following specifications for satisfactory No. 1, Crucible Flake<sup>a</sup>. For the sake of emphasis they are summarized here.

Dub suggested that a No. 1 flake is satisfactory, as regards carbon content when the content of graphitic carbon is not less than 85 per cent. Graphitic carbon is here intended to mean that carbon which remains after burning a dried sample three minutes at 800° C.

It is also suggested that 35 per cent of the product should remain on a 35-mesh standard screen, 30 per cent should pass through a 35-mesh and remain on a 65-mesh standard screen, and the remaining 35 per cent should be between 65 and 100 mesh. An allowance of 3 per cent through 100 mesh could be permitted.

The aim of the investigation herein reported was to produce material of the grade outlined from the concentrates of graphite from the Alabama graphite fields.

## PRESENT PRACTICE.

The production of crucible stock from the Alabama ores involves two distinct steps. The first is a concentration process by which a

---

<sup>a</sup> See page 40 of this bulletin for "carbon content" and page 39 for "screen analysis."

comparatively low-grade concentrate is recovered from the ore. The second step comprises the conversion of this low-grade concentrate into a high-grade material that will be satisfactory for crucible making.

The first step, being simply concentration, need not be discussed here. The second step, finishing, as it is called, is much more difficult to carry out and is the one that demands the greatest amount of investigation and improvement.

The present method of finishing the crude concentrate consists of, drying the concentrate, grinding the dried material in a buhr mill, and "bolting" or screening the ground product to remove the impurities that have been crushed during the grinding. The theory is that the graphite particles, being tougher, smoother, and thinner than the impurities, will be rubbed and polished between the buhr-stones with a minimum reduction in size, while the impurities themselves, owing to their more brittle characteristics and more massive structure, will be ground fine enough for removal by screening over a suitable cloth.

The method is simple, and in fact, within certain narrow limits and under favorable circumstances, gives good results. As a general rule, however, two difficulties are encountered that interfere with its commercial success—the grinding often destroys a large proportion of the coarse and valuable flake and there is a lack of uniformity in the finished product. The primary object of the present investigation was to determine the most efficient manner of obtaining the desired results and eliminating the troubles in the method mentioned above.

### CONCENTRATES USED IN TESTS.

As has been suggested, the problem involved is the treatment of the concentrates that are being produced in the Alabama district. Therefore, all of the work done during this investigation was on a series of representative samples from that district.

The concentrates produced in the Alabama district differ widely in their physical characteristics and graphitic content. However, as the physical properties of the different minerals composing the concentrates can not vary to any considerable extent, a scheme worked out for any one concentrate can be applied, with only slight variations and adjustments, to the other concentrates. For this reason, and because of the numerous schemes tried in the limited time available for the work, it was impossible to conduct tests of all of the concentrates. Therefore, it was decided to select several of the most representative for investigation, and to determine the extent that the results obtained on these concentrates could be generalized and applied to all.



Table 1 gives the results of screen analyses and assays of the different samples selected.

TABLE 1.—*Results of screen analyses and assays of different samples selected for experiment.*

Mesh.	Percentage by weight.	Carbon assay (per cent).	Units of carbon, each size. <sup>a</sup>	Percentage of total carbon.
<b>Sample 1:</b>				
14.....	0.1			
20.....	1.1	74.8	82.0	1.82
28.....	5.4	60.8	328—	7.27
35.....	14.8	53.6	793.0	17.60
48.....	22.5	47.2	1,062.0	23.55
65.....	21.8	44.4	968.0	21.47
100.....	17.4	44.00	766.0	16.99
150.....	8.8	43.40	382.0	8.47
200.....	2.7	41.20	111.0	2.46
—200.....	4.5	3.60	16.0	.35
Total.....			4,508.0	99.98
<b>Sample 2:</b>				
14.....	1.8	6.1	11.0	.21
20.....	2.4	15.7	37.7	.74
28.....	5.1	38.3	195.4	3.81
35.....	10.2	51.9	529.0	10.32
48.....	20.6	56.7	1,169.5	22.81
65.....	22.6	58.2	1,313.0	25.61
100.....	24.7	55.2	1,361.5	26.55
150.....	8.8	46.7	407.0	7.94
200.....	1.4	46.8	65.7	1.28
—200.....	2.3	16.2	37.2	.73
Total.....		51.27	5,127.0	100.00
<b>Sample 4:</b>				
14.....	Tr.			
20.....	Tr.			
28.....	.5	24.0	12.0	.37
35.....	6.2	20.7	128.0	3.92
48.....	16.1	25.9	417.0	12.79
65.....	23.1	33.1	765.0	23.46
100.....	25.7	43.4	1,114.0	34.16
150.....	16.1	49.1	780.0	23.92
200.....	4.0	56.1	22.0	.67
—200.....	6.5	36.9	23.0	.71
Total.....			3,261.0	100.00
<b>Sample 5:</b>				
14.....	.4			
20.....	1.8			
28.....	5.5	52.8	290.4	4.26
35.....	14.3	69.6	995.3	14.60
48.....	25.7	72.4	1,860.7	27.29
65.....	26.3	73.4	1,930.3	28.32
100.....	21.0	67.4	1,415.4	20.76
150.....	4.5	64.8	291.5	4.29
200.....	.3			
—200.....	.9	36.0	32.4	.48
Total.....			6,816.0	100.00
<b>Sample 6:</b>				
14.....				
20.....	.4			
28.....	2.1	60.2	126.42	2.98
35.....	7.7	53.6	412.72	39.82
48.....	13.4	48.2	790.48	18.27
65.....	22.4	45.2	1,012.48	23.82
100.....	24.5	35.6	872.20	20.55
150.....	17.0	38.4	652.80	15.42
200.....	4.0	39.0	156.00	3.68
—200.....	5.5	40.4	222.20	5.35
Total.....			4,245.30	99.89

<sup>a</sup> Column 2 × column 3.

The characteristics of the samples were briefly as follows:

No. 1. A fairly coarse concentrate assaying 48.95 per cent carbon.

No. 2. A coarse flake concentrate assaying 56.47 per cent carbon.



No. 4. A fine flake concentrate assaying 42.38 per cent carbon.

No. 5. A medium coarse flake concentrate assaying 73.30 per cent carbon.

No. 8. A medium coarse flake concentrate assaying 42.45 per cent carbon.

As the work progressed, it was found that duplication of all of the necessary experiments on even this number of samples would involve too much time; hence two of the most representative were selected for investigation, and the conclusions drawn from the results obtained from them were applied to the rest. The two samples chosen for the work were Nos. 2 and 4, as they represented two general classes of concentrate—a fine and a coarse medium-grade material.

### PHYSICAL PROPERTIES OF GRAPHITE AND ASSOCIATED GANGUE MINERALS.

It is evident that production from the concentrates described of a graphite stock containing 85 per cent or more carbon would first involve separation of the graphite from its contained impurities. This can be accomplished only by taking advantage of differences between the physical properties of the graphite and of the associated impurities. For comparison of the physical properties that might be utilized for this purpose they have been tabulated in Table 2, as follows:

TABLE 2.—*Physical properties of graphite and of associated minerals.*

Mineral.	Specific gravity.	Hardness.	Brittleness.	Shape.	Color.	Electroconductivity.	Surface condition.	Cleavage.
Graphite.	2.1	1 to 2	Not brittle.	Flat, tabular plates.	Dark gray and black.	Good....	Metallic luster.	One direction.
Quartz...	2.66	7	Very brittle.	Massive, granular, or compact.	Colorless, or may be any color.	Nonconductor.	Glassy or vitreous.	Practically absent.
Mica.....	2.8	2 to 2½	Not brittle..	Thin, scaly plates.	Colorless, or pale green to brown to black.	...do....	Smooth, glassy, or vitreous.	Nearly perfect in one direction.

Some of these properties vary greatly, but others are closely alike. Unfortunately the property of specific gravity, most often used in the separation of minerals, can not be used in finishing graphite because the weights of the various substances involved are nearly the same. On the other hand, properties that are the least often used in the commercial separation of minerals, for instance, electrical conductivity, are the properties that show the greatest amount of difference and must be depended upon to give the best results.

Thus the standard equipment manufactured for the separation of minerals, such as gravity tables, can not be used to the best advantage in finishing graphite. In fact, past experimental work with specific

gravity machines has substantiated this conclusion. Also, because no satisfactory equipment has been developed to make a separation by taking advantage of the little used physical characteristics, it was necessary to construct special apparatus.

### ASPIRATOR TESTS.

Each apparatus investigated during this work uses the physical properties or characteristics of the graphite and gangue minerals as a means to accomplish separation.

The first experiments were made to determine the possibility of separating the graphite and gangue material by differences in specific gravity and shape of the various particles. The apparatus first tried depends on the difference in specific gravity and the difference in shape of the various minerals to accomplish separation. This type of apparatus is already being used to a limited extent in Alabama and is there known as an "aspirator."

The aspirator is fundamentally an air classifier. This name is appropriate for the reason that the separation depends on the settling rate of the different minerals in a current of air.

The apparatus (Pl. I, A) constructed for the experiments was closely similar to those that are being used in the Alabama field at the present time. The material to be treated is fed from a hopper to a roller from which it drops into the large end of a long, sloping, horizontally placed box. The roll feeder is arranged so as to feed the material in an even curtain across the whole opening. From the small end of the box a pipe leads to a suction fan. The box is 13.5 inches wide by 12.5 inches high at the feed end, and 43.5 inches long. The suction pipe is 31 inches long and 4 inches in diameter.

Across the opening of the box and parallel to the ground line are four vanes or sand deflectors, placed at such an angle from the vertical that any material fed from the roller when the air current is off will be deflected to the outside of the box.

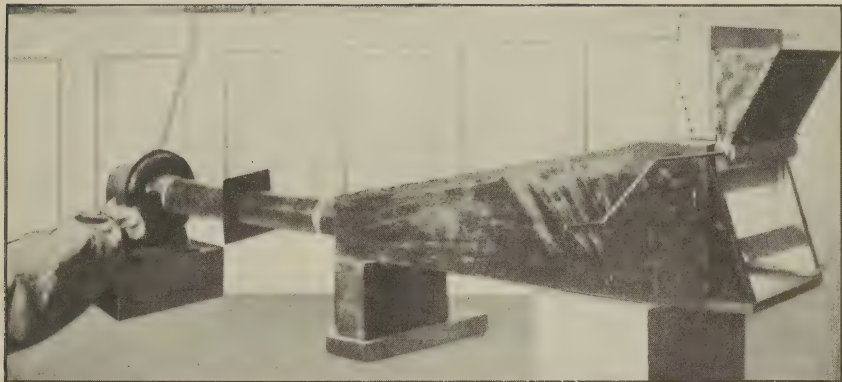
For a description of the operation of the classifier see page 25.

Several samples of the concentrates already described were tested with this machine, representative results of several runs being as follows:

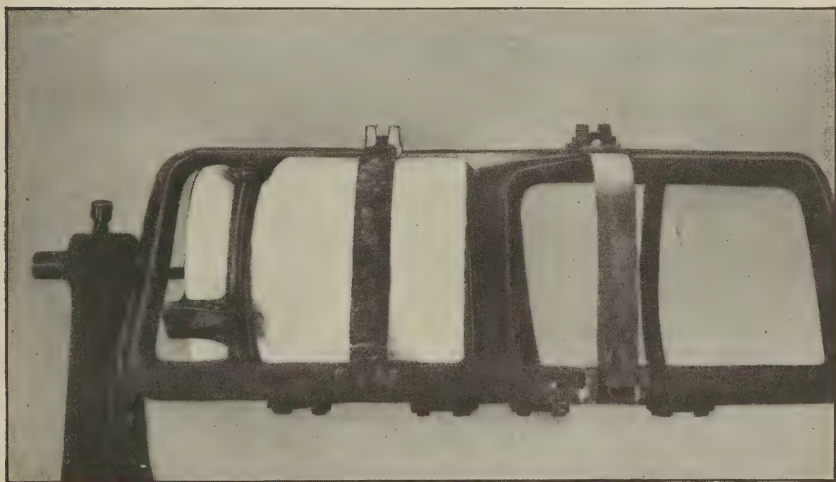
	Graphite, per cent.
Heads -----	51.23
Concentrates -----	69.23
Tailings -----	19.08

These results show a calculated recovery of 86.25 per cent of the total graphite in the concentrates produced.

The aspirator yields three products, as follows: Sand tailings, which are recovered on the outside of the box below the feed hopper; coarse graphite concentrates, which are heavy enough to be carried into the box and are deposited there, and the pure, fine concentrate



A. IMPROVISED ASPIRATOR USED IN TESTS AT SALT LAKE CITY STATION.



B. BUHR MILL USED IN GRAPHITE GRINDING TESTS.





that is too light to be deposited within the box, but is sucked through the blower and caught in a bag at its outlet.

In the particular test described, 87.1 per cent of the concentrate was deposited in the box and the remaining 12.9 per cent was drawn through the fan. The former material assayed 68.4 per cent graphite; the fine material assayed 74.85 per cent.

In order that the difference in the sizes of the various products made may be compared with the material before treatment, and with each other, the results of screen analysis and of assay of each of the sizes are given in Table 3 following. In this table the results for fine and coarse concentrates were combined.

TABLE 3.—Results of screen analyses and of assays of aspirator concentrates.

COARSE CONCENTRATES.

[87.1 per cent of aspirator concentrates.]

Mesh.	Proportion by weight.	Carbon assay.	Carbon, units. <sup>a</sup>	Total carbon.
	<i>Per cent.</i>	<i>Per cent.</i>		<i>Per cent.</i>
14.....	Tr.			
20.....	0.4	83.7	33.5	0.49
28.....	2.6	89.0	231.8	3.39
35.....	10.6	87.4	926.5	13.55
48.....	24.4	80.8	1,972.0	28.83
65.....	25.4	69.0	1,753.0	25.63
100.....	28.3	62.0	1,756.0	25.68
150.....	7.1	24.4	173.2	2.53
200.....	.5	23.3	11.5	.17
-200.....	.5	23.4	11.5	.17
Total.....	99.8	68.39	6,839.0	100.44

<sup>a</sup> Column 2 times column 3.

FINE CONCENTRATES.

[12.9 per cent of aspirator concentrates.]

14.....	None.			
20.....	None.			
28.....	Tr.			
35.....	1.6	88.1	141.00	1.70
48.....	9.2	90.3	831.00	10.92
65.....	24.1	87.8	2,118.00	28.11
100.....	37.8	81.1	3,063.00	40.74
150.....	17.1	62.8	1,073.00	14.17
200.....	3.9	48.8	190.00	2.36
-200.....	5.8	29.2	169.00	2.08
Total.....	99.5	74.85	7,585.00	100.08

ASPIRATOR TAILINGS.

14.....	5.00	4.1	20.5	1.05
20.....	8.00	12.9	103.2	5.41
28.....	12.80	23.2	297.5	15.59
35.....	18.50	26.9	498.0	26.10
48.....	28.40	24.0	681.5	35.69
65.....	20.00	13.2	264.0	13.83
100.....	6.40	5.9	37.8	1.98
150.....	.20	12.3	2.5	.13
200.....	Tr.			
-200.....	.20	15.3	3.0	.15
Total.....	99.5	19.08	1,908.0	99.90

It should be noted that the greatest concentration and improvement of product is in the coarser material of the coarse concentrates. The great difference between the carbon content of the material coarser than 150 mesh and of that finer than this size is very striking and also important.

As regards the fine concentrates produced the large percentage (83.02), which is -48 mesh and +150 mesh, is noteworthy as is the high carbon content of the +200 material.

Treatment of another sample, No. 4, in the aspirator gave the following results, showing a calculated recovery of 96.2 per cent of the total graphite in the concentrates produced:

	Carbon, per cent.
Heads-----	42.61
Concentrates-----	56.62
Tailings-----	6.03

These results indicate that although some of the impurities can be removed with the aspirator, thereby materially raising the grade of the material, the application of this type of aspirator will never be general.

It has long been known that the coarse sand particles cause the greatest destruction of the large flake in the buhr mill. When the concentrates carry a large proportion of sand particles of such a nature that the type of treatment described will raise the grade of the crude product to 70 per cent or better, the aspirator can undoubtedly be used to great advantage before treating the concentrates in the buhr mill. However, unless there is a comparatively large proportion of such impurities in the crude concentrate, the aspirator can not be used to the best advantage. The apparatus should certainly not be incorporated in a finishing mill except after thorough and comprehensive tests to determine the results that can be obtained by its use, and then only on the advice of a competent engineer conversant with the conditions that must be met by the installation.

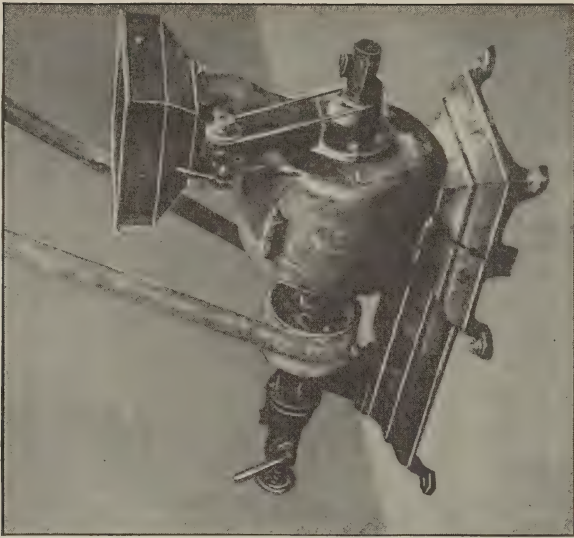
Probably the most potent reason for the comparatively poor results obtained with the aspirator was the wide variety in size of the material treated. When a current of air strong enough to remove the coarse graphite flakes from the feed was used, the fine sand would, at the same time, be drawn into the box with the concentrate.

However, when the concentrates are of low grade and contain a large quantity of coarse sand, treatment by such a simple process as the one just described is so advantageous that its use should be considered, and when feasible, the designing engineer should include an aspirator in a projected plant.





A. PNEUMATIC JIG USED IN TESTS.



B. BUHR MILL USED IN TESTS.



## TESTS WITH PNEUMATIC JIG.

The pneumatic jig was the next apparatus tested. Its construction differs greatly from that of the aspirator, although the principles involved in the operation of each are similar. As this equipment has been manufactured on a commercial scale for several years, its use would be advantageous when feasible, as a more simple and standard "layout" could be arranged than where improvised equipment like that described was used. Both types would be simple to operate, the aspirator being capable of more sensitive adjustment.

The jig used in the graphite tests is a small laboratory apparatus, built by the manufacturers of the standard machine, for laboratory experiments. Plate II, 4, shows the jig as set up for work. It comprises a small metal box, the bottom of which is made of a screen 3.25 by 2.5 inches. Air pressure of 2 to 4 ounces, furnished by a pressure blower or compressor is admitted to the bottom of the box under the screen through a small rapidly rotating valve. The rapid pulsation of the air, due to its passage through the valve, causes the material on the screen to remain in continuous agitation and partial suspension. Consequently, the coarse sandy impurities, being heavier, collect at the bottom, the lighter graphitic material being stratified above the sand. The jig is so arranged that operation is continuous, the sand being constantly collected and discharged from one side, while the lighter graphite is separated above the sand and discharged from the top of the column and on the other side.

Several tests were made with the jig, representative results being as follows:

*Results of two representative tests with pneumatic jig.*

	Carbon, per cent—	
	Sample 2. <sup>a</sup>	Sample 4. <sup>b</sup>
Heads -----	51.23	42.61
Concentrates -----	69.66	64.13
Tailings -----	15.11	3.14

It is seen that the recovery of clean tailings from the treatment of sample 4 was higher than from the treatment of sample 2. In order that the products after treatment can be compared with the original feed as to size and carbon content, the screen analysis and related data are given in Table 4 following:

<sup>a</sup> A calculated recovery of 91.25 per cent of the graphite in the jig concentrate is indicated.

<sup>b</sup> A calculated recovery of 97.5 per cent of the graphite in the jig concentrate is indicated.

TABLE 4.—*Results of screen analyses and of assays of jig concentrate and tailing.*

[Sample 4.]

Mesh.	Concentrate.				Tailing.			
	Proportion by weight.	Carbon assay.	Carbon, units, <sup>a</sup>	Total carbon.	Proportion by weight.	Carbon assay.	Carbon, units, <sup>a</sup>	Total carbon.
	<i>Per cent.</i>	<i>Per cent.</i>		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>		<i>Per cent.</i>
14.....	None.				0.1	5.2		
20.....	Tr.				.4		2.1	0.66
28.....	0.4	80.9	32.5	0.51	2.6	5.7	14.8	4.63
35.....	3.1	78.4	242.5	3.78	15.7	4.0	62.8	19.65
48.....	10.5	76.4	802.5	12.51	28.6	3.4	97.2	30.42
65.....	20.8	69.7	1,450	22.61	27.8	3.1	86.2	26.98
100.....	29.1	64.5	1,875	29.23	18.7	2.6	48.6	15.21
150.....	21.1	62.6	1,322	20.61	9.8	.8	7.8	2.44
200.....	5.8	55.1	319.5	4.98	.2			
—200.....	9.3	39.8	370	5.77	.3			
Total.....	100.1	64.12	6,414	100.00	99.2	3.19		

<sup>a</sup> "Proportion by weight" times "carbon assay."

The results given in the table indicate that the air jig has practically the same field of application as the aspirator, also that it can be made to produce a cleaner tailing than the aspirator, but the aspirator will give a higher grade of finished product. This condition will not, however, interfere with the jig's usefulness, provided the material it removes from the crude concentrate is the coarse sand that causes the high loss of coarse flake in the buhr mill. An inspection of the various products of the jig will indicate that it does remove the coarse sand particles.

Many of the remarks regarding the use of the aspirator apply also to the air jig. There is no question that under certain circumstances the jig can be made to do valuable work in the preparation of crude graphite for crucible making, but the character of the crude concentrates themselves, and the other factors that enter into the commercial finishing of concentrates must be considered before the jig be recommended for use in preliminary treatment. If the jig is used only to do that work for which it is particularly adapted, the removal of coarse sand impurities, its success is assured.

### TESTS WITH PEBBLE MILL.

Probably the most striking characteristics of graphite flakes are their toughness and shape. In these two respects the flakes differ greatly from the impurities usually associated with the graphite in the crude concentrates. It was hoped, therefore, that these properties could be utilized in separating the graphite from the associated impurities.

It was thought that the coarse sand impurities, although much harder than the graphite flakes, were so much more brittle that the concentrate could be so ground as to reduce materially the size of the sand particles without causing a material reduction in the size of the graphite particles. A careful survey of the different classes of grinding machinery indicated that this operation could be carried out with greatest chance of success in a mill of the ball or pebble type.

The difference in the specific gravities of the two materials was also a reason for believing that this sort of selective grinding could be done. It was thought that under the correct conditions of dilution the heavier sandy material would tend to classify in the bottom of the charge among the grinding balls where it would be ground more than the graphite, which, because of its lighter weight would tend to segregate higher in the charge and out of the grinding zone. Such a separation corresponds closely to that in a hydraulic classifier. The results obtained from experiments proved that in most particulars the assumptions were fundamentally correct.

After a treatment such as suggested, which would produce a product with the sandy constituents ground finer than the flake graphite, two methods of separating the sand and graphite are available.

One of the methods is by screening. Experiment showed that after such grinding, the grade of the original material could be raised 25 per cent or more by simply passing the ground concentrate over a 100-mesh screen. It will be recalled that 100-mesh material is the smallest size that can be satisfactorily used in crucible making.

The other method is by flotation, the process by which much of the crude concentrate produced in the Alabama field is made. Tests revealed that in order to recover the coarse flake in the ore by flotation, it was necessary to make a fairly low grade concentrate. The reason seemed to be that some of the silicious material was porous and sponge-like in structure, and contained quantities of entrained air which invariably caused the sand to float with the carbon.

Wet grinding of the concentrate in a ball mill tended to break up the pumice-like silica and to eliminate the entrapped air. When this had been done, the silica did not float as readily as before. Therefore, after grinding, the concentrate can be refloatated and its grade raised to a degree corresponding to the amount of the spongy sand removed.

A small laboratory pebble mill (see Pl. I, *B*, p. 50) was used in the grinding tests. The charges consisted of 50 per cent water and 50 per cent crude concentrate. The grinding periods varied, but at first were of 30 minutes' duration.

A certain sample contained 90 per cent of the carbon as material coarser than 100 mesh which assayed 53.9 per cent carbon before grinding. After grinding in the pebble mill, 81 per cent of the carbon still remained on a 100-mesh screen as material which assayed

61.3 per cent carbon. These results indicated that a decided improvement had been made in the grade of the material coarser than 100-mesh, with a loss of only 9 per cent of the valuable coarse flake.

On the strength of these results, it was decided to make a series of tests under the conditions already described, but varying the length of the grinding period in each test. The results of this series of experiments are given in Table 5 following:

TABLE 5.—Results of grinding graphite samples in pebble mill with equal parts of water.

	Time of grinding (hours).	Grade of concentrate (percentage of carbon in +100-mesh material).	Carbon remaining on 100-mesh screen.	
			Percentage of total carbon of original sample.	Percentage of +100-mesh carbon in original sample.
Sample 2.....	0	53.9	90.00	100.00
	$\frac{1}{4}$	61.3	81.00	90.90
	$\frac{1}{2}$	77.6	80.60	89.60
	1	81.7	83.30	92.50
	$1\frac{1}{2}$	79.8	81.40	90.40
	$1\frac{3}{4}$	81.4	72.60	80.60
	2	82.1	67.40	74.80
Sample 4.....	0	47.9	80.5	100.00
	$\frac{1}{4}$	70.6	65.9	81.80
	$\frac{1}{2}$	74.6	62.7	77.90
	1	81.4	62.5	77.60
	$1\frac{1}{4}$	75.4	58.7	72.90
	$1\frac{3}{4}$	80.5	59.3	73.60
	2	82.4	51.1	63.50

The results are plotted in figures 16 and 17. The figures show that the correct time for grinding, under the conditions of this set

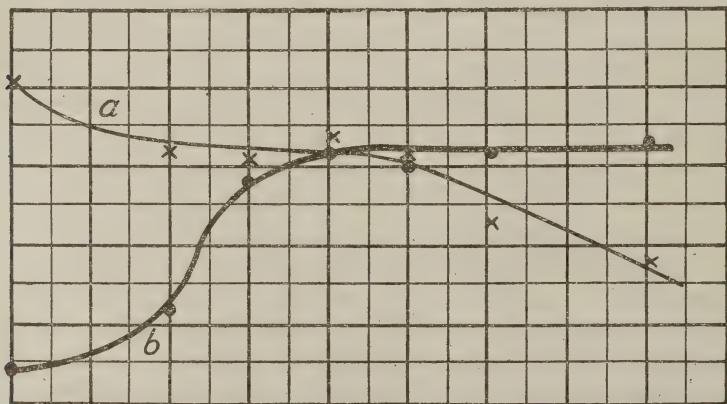


FIGURE 16.—Curves showing results of grinding sample 2 in pebble mill with equal parts of water. *a*, Recovery; *b*, carbon

of experiments, is the time that will produce the highest grade of +100-mesh material with a minimum destruction of the flake that originally existed in the concentrates coarser than 100 mesh.



On the curve this point will be at the intersection of the line representing the assay of the +100-mesh material with the line representing the recovery of +100-mesh carbon. For sample 2 the time is about one hour, whereas for sample 4 it is about 30 minutes.

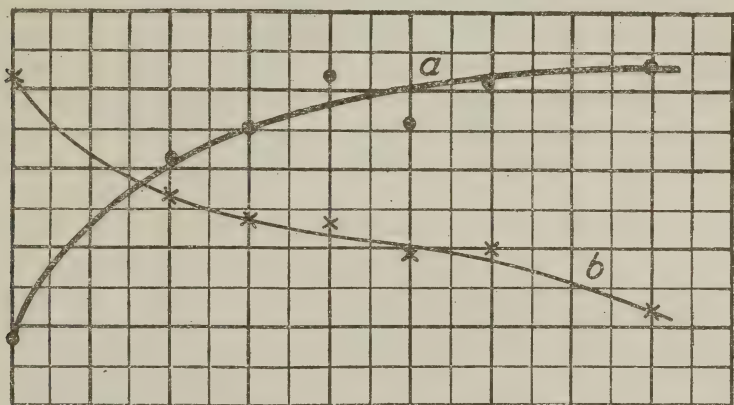


FIGURE 17.—Curves showing results of grinding sample 4 in pebble mill with equal parts of water. *a*, Carbon; *b*, recovery.

It is evident from these results that the conditions that give the best results will vary with the physical characteristics of the crude concentrate itself, and must be determined by actual experiment with the material that is to be treated in the commercial plant.

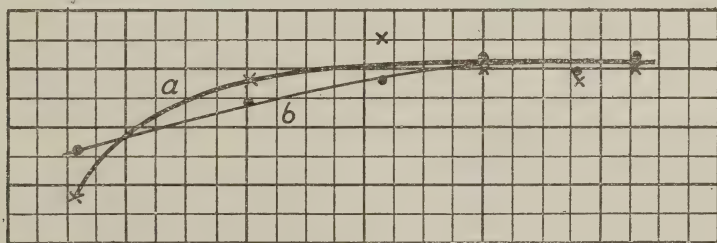


FIGURE 18.—Curves showing results of grinding sample 2 in a pebble mill with different percentages of water. *a*, Recovery; *b*, carbon.

In order to determine the effect of varying the amount of water in the pebble-mill charge, several tests were made, with sample 2, the other conditions being similar to those that had been proved to be the most satisfactory in the previous series. The results of these tests are shown in Table 6 following and in figure 18. The concentrates were fed to the mill at the rate of 400 grams per hour.

TABLE 6.—Results of grinding graphite sample 2 in pebble mill with different percentages of water.<sup>a</sup>

Quantity of water used (c. c.).	Grade of concentrate (percentage of carbon in +100-mesh material).	Carbon remaining on 100-mesh screen.		Water in mill pulp (per cent).
		Percentage of total carbon of original sample.	Percentage of +100-mesh carbon of original sample.	
0	53.9	90.00	100.00	0.0
300	77.6	73.7	81.90	42.8
400	81.7	83.3	92.50	50.0
500	83.5	87.0	96.60	55.55
600	85.0	84.50	93.30	60.0
700	84.1	83.40	92.50	63.6
800	84.7	85.10	94.50	66.6

<sup>a</sup> See also figure 18.

Figure 18 shows that no improvements in the results are to be noted after a dilution of 60 per cent water is reached. In this experiment the grade of the +100-mesh material was raised from 53.9 per cent carbon to 85 per cent carbon, with a loss of only 7 per cent of the carbon that was coarser than 100 mesh before grinding.

#### DISCUSSION OF PEBBLE-MILL RESULTS.

In summarizing the results obtained by the pebble-mill grinding, it may be said that the apparatus, when operated under the conditions that have been determined to be the most satisfactory, will raise the grade of the +100-mesh material in a crude concentrate to such a degree that the following buhr-mill grinding can be carried on to give the maximum results possible with it. At the same time, the results indicate that the pebble mill can not be used to produce a finished grade of product. In other words, the pebble-mill grinding should be used only as a step in the process, and preparatory to treatment in the buhr mill.

The choice of method of removing the impurities that are ground in the pebble mill will depend on the character of the concentrates that are being treated and other conditions that will be encountered in the commercial finishing plant.

The results obtained in the experiments here described indicate clearly that pebble-mill grinding has a wide field and can be made to produce favorable results even under the most adverse and difficult operating conditions.

#### TESTS WITH ELECTROSTATIC SEPARATOR.

The great difference between the electroconductivity of graphite and that of the principal impurities associated with it in the crude concentrates early suggested that a method of separation based on

electroconductivity might be developed. Such a method has already been tried to a certain extent in the Alabama field, but the results in general have been unsatisfactory.

However, in one part of the Alabama district producing concentrates high in mica but very low in iron, the process worked well on a commercial scale and is being used there at the present time. Such material is particularly adapted for electrostatic treatment; hence the process gives satisfactory results, although certain changes in the separator as originally designed were made in order to overcome inherent mechanical defects. As this condition was only local, however, electrostatic separation was never successfully used in the rest of the district.

Improvements in the concentration mill made possible the production of concentrate carrying so much less iron that it was thought that the iron would no longer be a determining factor, and that hence the electrostatic process should offer much more chance for success than previously. Thus, in order to use this method of separation, it seemed necessary only to construct a machine that would eliminate the mechanical difficulties previously encountered in the operation of this type of apparatus.

With this end in view, a series of experiments was made with a large machine of the type at present on the market, in order to determine definitely the reason for its previous failure. However, with this machine the separation was not satisfactory. The poles of the machine are so arranged that the graphite particles receive a certain charge from one pole and are thrown into the field of the opposite pole; then the charge first obtained is neutralized and the particles are thrown in the direction of the first pole. Consequently the material progresses through the apparatus in a zigzag path, first in one direction and then in the opposite, with the result that no separation of the graphite from the nonelectroactive particles is obtained. The sand and other impurities, not being affected by the charges acting on the graphite, fall directly through the machine. Therefore, the graphite will be separated from the impurities while passing the first pole, but on passing the oppositely charged pole will be thrown in the opposite direction and into the sand from which it has just been separated. Eventually the mixture is discharged practically unchanged from its original state.

The experiments demonstrated that in order to overcome the faults mentioned it would be necessary so to design a machine that when a graphite particle had received a charge of a certain sign, it would carry this charge unneutralized, and not be forced in the opposite direction and lose its tendency to separate.



Many experiments were made with this end in view, and two types of apparatus were constructed. It was found possible to build a machine of such type that only one set of poles would tend to impart a charge to the graphite particles; these would be repelled directly into the concentrate hopper and given no chance to be neutralized and thrown back into the impurities.

Figure 19 shows the principal details of the separator as finally constructed, and indicates the manner of its operation. A view of the separator as set up ready for work is shown in Plate III.

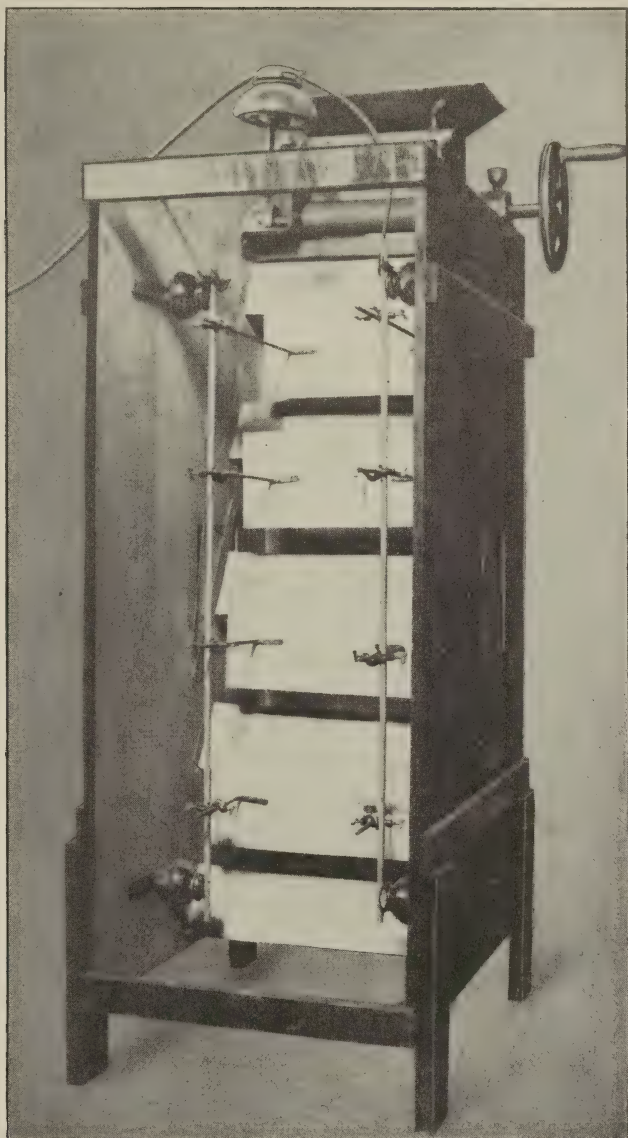
The material to be treated is placed in the hopper at the top of the apparatus and is fed from the hopper by a roller in the bottom. The graphite first strikes one of the inclined planes that carries a weak, negative charge. While the graphite is on this plane it therefore receives also a weak, negative charge. As the plane and the graphite both have the same charge, the graphite will be repelled and will jump from the negative vane into the field induced by the positive vanes. On coming into this opposite field, the graphite gives up its weak, negative charge and receives instead a much stronger positive one, which in turn causes it to be repelled violently from the positive plate and through the openings shown between the electrodes. After having passed to the back of the plates the graphite is free to fall directly into the concentrate hopper at the bottom of the separator.

The sand and mica, being much poorer conductors of electricity, receive such a weak charge from the negative plate on which they first fall that they are not repelled into the strong, positive field, but simply slide from one negative plate to the next and so on down through the machine until finally discharged into the tailings hopper at the bottom of the apparatus and in front of the concentrates hopper.

The electrical equipment used to operate the separator consisted of a 5-k. v. a. transformer designed and built as an oil-testing transformer. It is equipped with a hand-operated voltage regulator so arranged that the voltage can be varied from zero to the maximum. While the transformer has a secondary voltage of 50,000 volts, only half of this is available for rectification as the center of the secondary coil is permanently grounded to the transformer case.

The rectifier is of the mechanical type. It consists of a rotating disk driven by a synchronous motor. Stationary shoes and revolving electrodes on the disk make and break the circuit so as to rectify alternating waves. This arrangement gives a unidirection pulsating current necessary for the operation of the machine. Figure 20 shows the wiring diagram of this equipment.





ELECTROSTATIC GRAPHITE SEPARATOR.



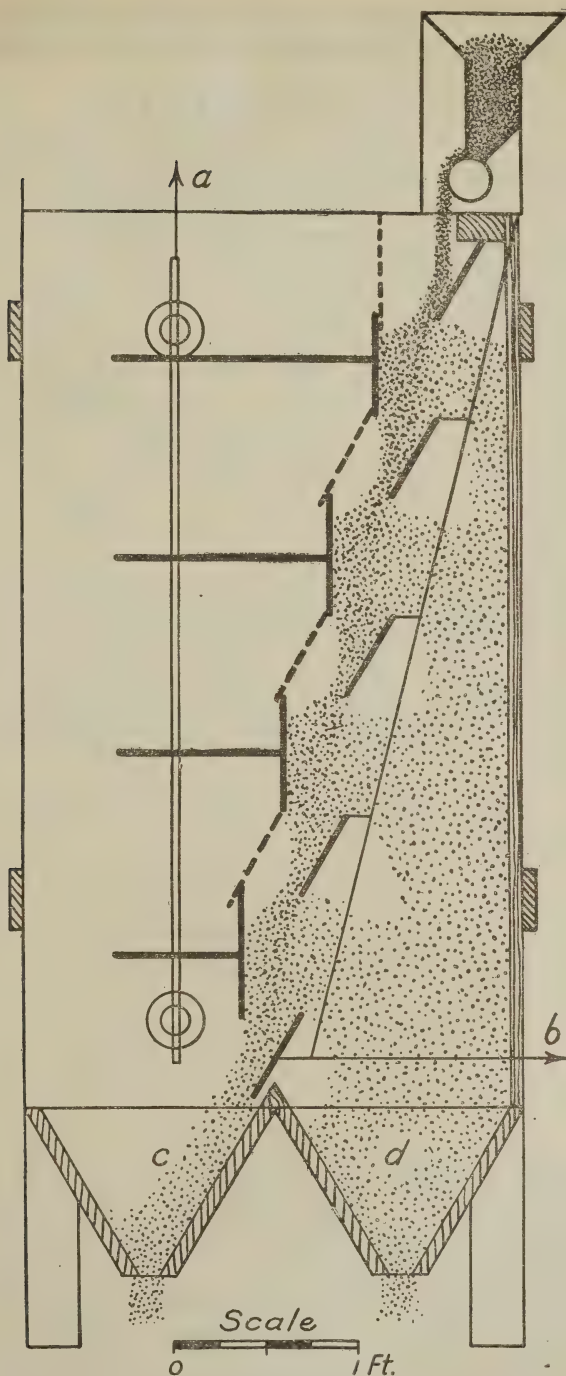


FIGURE 19.—Electrostatic graphite separator designed by Bureau of Mines. *a*, To rectifier; *b*, to ground; *c*, tailings; *d*, concentrate.

The table following gives a series of readings taken while the electrostatic separator was operated at the various voltages used in

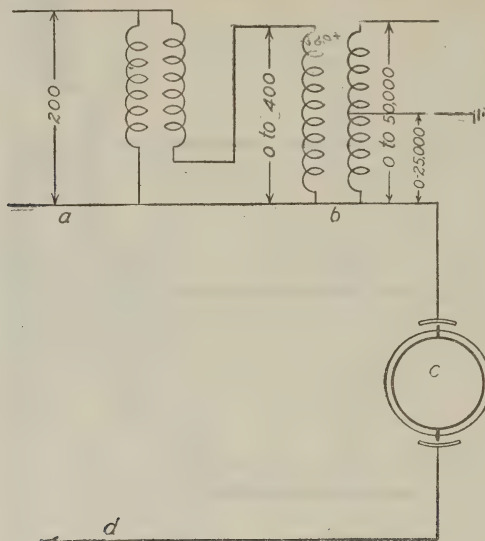


FIGURE 20.—Wiring diagram of electrical equipment used for electrostatic graphite separator designed by the Bureau of Mines.

testing. This table shows that only 200 watts is required to operate the apparatus at the highest voltage. This is equivalent to only 0.87 k. v. a.

*Results of transformer readings taken in electrostatic-separator tests.*

Primary coil.			Secondary coil.	
Volts.	Amperes.	Watts.	Power factor.	Volts.
446	2.22	2.12	0.21	27,900
440	1.95	2.00	.33	27,500
420	1.61	1.96	.29	26,250
388	1.15	1.52	.34	24,200
360	.90	1.32	.40	22,500
272	.50	.72	.53	17,000

The equipment (5 k. v. a.) is therefore large enough to operate a battery of six machines of the same type and size as the one described. Obviously the cost of power for the concentrating of graphite by electrostatic means will be small, an important consideration where power is costly.

After the completion of the equipment just described, a series of tests was made with the various concentrate samples in order to determine the efficiency of the device and the limitations of its use.



The voltage used in all of the tests was 28,500. The table following shows the results obtained in the treatment of the various samples.

*Results of tests with electrostatic separator.*

Sample No.	Percentage of graphite in—			Percentage of concentrates—	
	Feed.	Concentrates.	Tailing.	Coarser than 100 mesh.	Finer than 100 mesh.
1.....	48.95	59.02	25.16	61.5	24.2
2.....	51.25	74.23	13.90	75.8	59.8
2.....	51.23	78.81	19.04	80.2	61.6
4.....	42.61	68.58	8.30	71.1	24.7
4.....	42.61	75.89	-----	80.6	65.2
2-100.....	81.2	87.62	-----	88.5	60.4
Bohr mill:					
1.....	48.95	64.00	25.00	66.2	58.4
5.....	73.80	83.0	52.80	63.2	73.2
4.....	42.60	71.20	15.60	71.6	71.0
8.....	42.45	67.20	17.80	77.4	62.4
9.....	40.44	80.40	33.00	81.6	72.4

The results obtained with the electrostatic separator can be considered most satisfactory for two reasons. The tabulated results show that in every test the grade of the concentrates was considerably higher than the material fed to the machine. This improvement was accomplished without grinding, a step that will, in spite of all precautions, cause a certain reduction in size of the original coarse material in the crude concentrates.

In each of the eleven tests, except one, the concentrates were increased in grade by 10 to 40 per cent carbon. The proportion of carbon remaining in the tailings of the electrostatic separator varied greatly, but this variation was due, in almost every instance, to the carbon that is too fine to be acted upon by the apparatus in the most efficient manner, or to the variation in the number of flakes of graphite attached to pieces of sand large enough to cause them to be drawn into the tailings. Obviously the degree of variation will depend on the condition of the concentrates themselves and not on the treatment to which they are subjected. The photomicrographs presented in connection with the discussion of flake structure on pages 66 to 67, show this condition clearly.

As the value of graphite flake for crucible making depends on the assay of the material that is coarser than 100 mesh, the value of any plan of treatment will depend on the grade of this 100-mesh material produced. For this reason, the assays of the 100-mesh part of the concentrates are given, as well as the assays of the whole product. The results of the +100-mesh assays show that out of the eleven tests reported only two of the +100-mesh concentrates contained less than 70 per cent carbon, while seven of the number contained more than 75 per cent graphite. Such results were especially

satisfactory in view of the fact that the concentrates had not been ground, with the consequent loss of coarse flake always resulting from a treatment of this nature.

It was found necessary to give the concentrates two treatments in the electrostatic separator in order to obtain optimum results. This requirement could be met in a commercial installation in one of two ways—by installing twice the number of separators required for a single treatment or by constructing apparatus containing twice the number of poles in the experimental equipment. It is probable that the latter method would prove the more satisfactory.

#### SUMMARY OF RESULTS WITH ELECTROSTATIC SEPARATOR.

The results obtained with the electrostatic separator prove conclusively that electrostatic separation can be successfully used in the finishing of graphite. They also indicate that the separators on the market at present will have to be changed in certain respects in order to make possible the obtaining of satisfactory results by their use. The results also seem to prove that the apparatus has a wider application than is possible with any of the other devices tested, with the possible exception of the buhr mill.

The electrostatic process seems also to be the most economical, as the cost of power is low and the loss of coarse and valuable flake is small.

At the same time, it is to be expected that, although the device will invariably improve the grade of the concentrates treated, it will not yield a finished product containing 85 per cent carbon, which is satisfactory for crucible manufacture.

However, the improvement in grade of the concentrates tested was so great as to indicate that electrostatic separation will, with few exceptions, produce a feed for the buhr mill that will be ideal for that apparatus to finish to the crucible requirements without excessive loss of coarse flake. The removal of the coarse sand was almost complete.

The results already given show that although the electrostatic separator will benefit the concentrates the degree of beneficiation will vary. Seemingly, the degree of variation depends on the physical condition of the concentrates themselves previous to treatment. In other words, the results to be obtained with this separator, like those with any other type of equipment, will depend on the concentrates themselves. Hence there can be no doubt that conditions will sometimes be met under which this type of treatment would not be as satisfactory as some other that might be developed for the removal of the coarse sand. However, the process has a more general application than any of the methods described thus far.

## FLOTATION AS A FINISHING PROCESS.

Flotation as a process of concentrating the graphitic content of ores has been used for some time and with a fair degree of success. For this reason and because graphite has a much greater tendency to float than the impurities that are usually found associated with it the idea of refloating the concentrate to remove the impurities before buhr-mill treatment was suggested.

There are two primary reasons for thinking that double flotation could be used in this connection. The first is the law of concentration that the grade of the concentrates produced by a concentration process will vary directly with the richness of the feed being treated. As it is possible to produce a 45 per cent graphite concentrate from ore carrying only 3 per cent graphite, it is reasonable to believe that, under the proper conditions, it should be feasible to make a concentrate carrying, say 70 per cent or more of graphite from a feed carrying 40 per cent or more of graphite.

The other possible reason for the success of flotation has already been suggested in the discussion of the use of the pebble mill. The point in mind is the porous condition of much of the silicious material that is found in the crude flotation concentrates. It was believed that if the concentrates from the first flotation treatment were to be ground or given some other treatment by which the sand could be reduced in size sufficiently to liberate the entrained air this siliceous material could be largely removed by a second flotation treatment.

## FLOTATION TESTS.

In order to prove or disprove these assumptions, three tests were made with a sample of flotation concentrate. One of these tests was made after the sample had been ground in the buhr mill, one after a treatment in the pebble mill, and one with material just as received without any treatment preliminary to the refloating.

The results obtained by this series of tests were as follows:

*Results obtained in three refloatation tests.*

Sample No.	Previous treatment before refloating.	Graphite in—			Recovery.
		Heads.	Concentrates.	Tails.	
		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
2.....	None.....	51.23	67.35	10.8	93.7
2.....	Buhr mill.....	51.23	80.73	2.1	98.5
2.....	Pebble mill.....	51.23	83.40	5.6	97.7

These results make it evident that simple refloatation, without a preliminary treatment, will materially raise the grade of the prod-



uct, and give a high recovery of the carbon content of the feed. However, as would be expected, the results do not indicate that simple refloating would be satisfactory in commercial practice.

As regards refloating following treatment in buhr or pebble mills, the results are such as to justify its adoption in a commercial installation.

Both the mechanical and the pneumatic types of flotation machines were used in the tests and, as far as could be noted, neither of the machines showed an advantage over the other.

Several oil mixtures were used. A mixture of 2 pounds of No. 17 oil and 0.75 pound of No. 5 oil, both made by the General Naval Stores Co., was found to be satisfactory in each test.

A series of tests was also made with sample 2 in order to determine the different results that could be obtained with different oil mixtures and various periods of agitation. These results are added for general interest, and to show what poor results can be obtained from a sample, by inefficient operation and incorrect conditions, as compared with the satisfactory results possible under proper conditions.

*Results of refloating crude concentrates.*

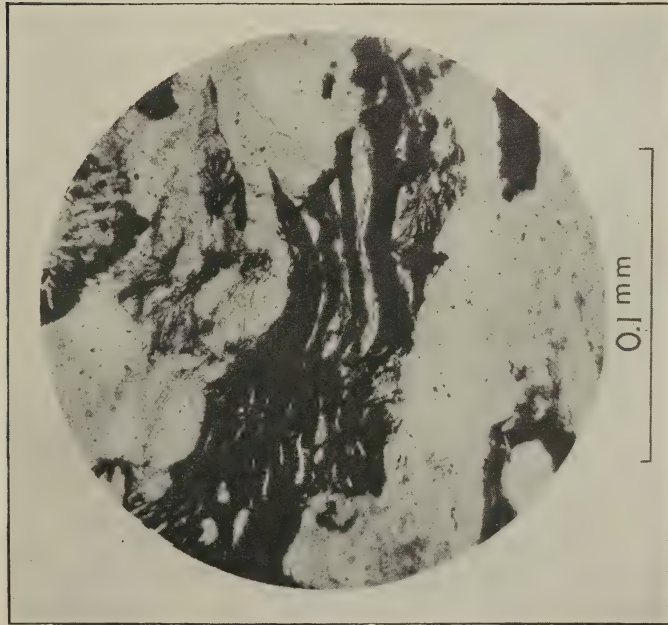
Sample No.	Preliminary treatment.	Percentage of graphite in—			Assay of concentrates.	
		Heads.	Concentrates.	Tails.	Coarser than 100 mesh.	Finer than 100 mesh.
2.....	Buhr mill.....	51.23	84.24	39.3	88.60	70.5
4.....	do.....	42.61	69.09	2.5	73.9	63.1
2.....	do.....	51.23	80.73	2.1	82.7	67.6
2.....	Pebble mill.....	51.23	83.10	.....	86.0	77.1
2.....	None.....	51.23	67.35	10.8	68.4	59.3
2.....	Pebble mill.....	51.23	87.6	7.5	.....	.....
2.....	do.....	51.23	76.27	17.68	78.6	49.0
2.....	do.....	51.23	83.4	5.6	.....	.....

In discussing the use of flotation, it might be said that the process when used to treat concentrates that have been previously ground, for instance in a pebble mill, will produce a material that will be particularly amenable to the final treatment in a buhr mill. The results indicate that supplementary treatment will be necessary to produce material containing the required 90 per cent graphite.

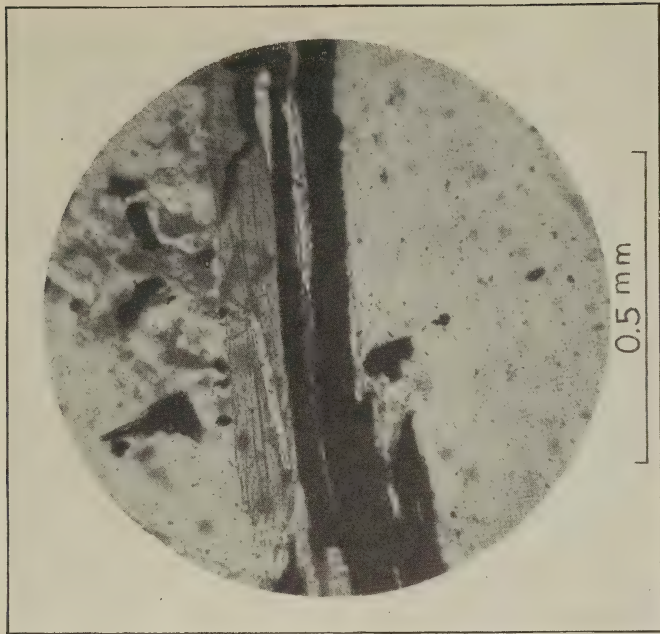
### THE BUHR MILL; ITS PLACE IN GRAPHITE FINISHING.

The function of the buhr mill has been previously discussed in this report. The action of the machine has also been suggested, with the two principal reasons for unsatisfactory results under certain conditions, namely, lack of uniformity in the product finished by it, and the excessive destruction of flake sometimes attending its use.

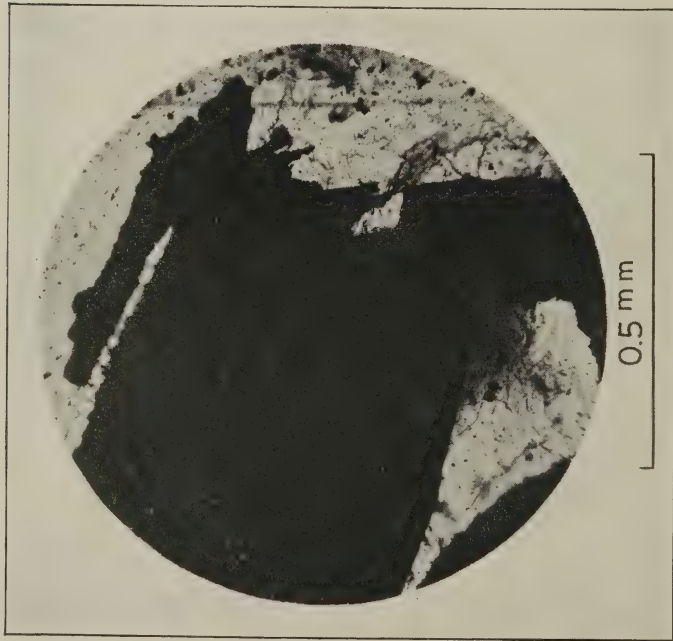




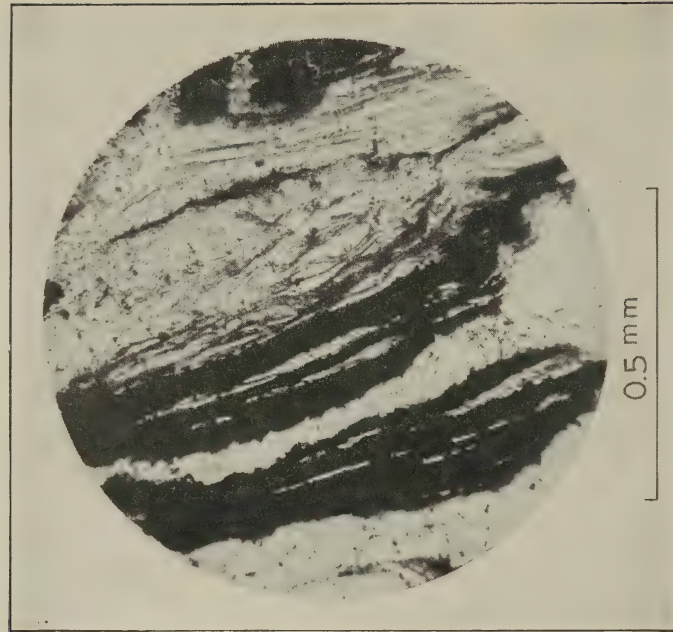
4. MICROPHOTOGRAPH OF GRAPHITE ORE, SHOWING  
IRREGULAR STRUCTURE.



B. MICROPHOTOGRAPH OF GRAPHITE ORE, SHOWING BANDED  
STRUCTURE.



4. MICROPHOTOGRAPH OF GRAPHITE ORE, SHOWING A LARGE  
PLATE WITH A TRIANGULAR INCLUSION.



B. MICROPHOTOGRAPH OF GRAPHITE ORE, SHOWING  
IMPURITIES EMBEDDED IN FLAKES.

During the comparatively long period that this machine has been used to finish graphite, methods of overcoming inherent difficulties in its use have become evident. One requirement is that the concentrate be of comparatively high grade before grinding in the buhr mill. With such material, no difficulty is encountered in consistently producing a crucible stock assaying 90 per cent graphite, provided the other conditions are correct.

In order that the excessive destruction of coarse flake incident to buhr mill grinding may be eliminated, it is necessary only to remove the coarse, gritty particles, or at least those coarse enough to require a large amount of grinding in the buhr mill.

Stated simply, when the buhr mill is used principally as a grinding machine on low-grade material, the grade of the product resulting can not be depended upon, and an excessively large proportion of the flake will be ground to dust. When, on the other hand, the machine is used only as a polishing or buffing machine, it can be trusted to give satisfactory results.

As already mentioned, the limitations of the mill were realized early in the progress of the work, and the experiments were conducted accordingly. Most of the products resulting from the various treatments contained less than the required 90 per cent of carbon and, therefore, could be used by the crucible makers only after a treatment in the buhr mill, to bring them to the necessary grade.

A view of the buhr mill used in the experimental work is shown in Plate II, *B* (p. 53).

#### RELATION OF FLAKE STRUCTURE TO FINISHING.

In order to show more clearly the truth of the assumptions previously stated, microscopic studies were made of graphite flakes, both before and after finishing. Also a series of microphotographs were made of this material. Four of these are reproduced in Plates IV and V.

The photographs show that the flakes were composed of series of thin laminations or plates of graphite, interbedded with thin pieces of quartz, mica, and other impurities. In some material these impurities were seen to be entirely surrounded by the carbon.

In order to determine the proportion of the impurities held in this manner, flakes of seemingly pure graphite, as it occurs naturally in the ore, were carefully picked by hand from the crude concentrate and assayed. In selecting the sample, special care was taken that nothing but the free, natural flakes were obtained for assay. The sample obtained in this manner was then carefully assayed, with the following results: Moisture, 1.26 per cent; carbon, 90.2 per cent; ash (impurities), 8.54 per cent. Examination of the ash



under the microscope showed that it consisted principally of mica with some quartz. This experiment indicated conclusively that impurities are imbedded between the laminations of the graphite, and also that the removal of all impurities not attached to the graphite flakes would give a finished product assaying only 90 per cent carbon.

This figure is, then, close to the limit of graphitic content possible in a finished material, even after treatment by a perfect finishing process.

Evidently the interlaminated impurities must be freed from the graphite flakes before they can be removed from the crude concentrate, and the only feasible method of accomplishing this is by separation of the graphite flakes into their individual laminations. It follows that the more there are of these interbedded layers to be removed, the greater will be the number of plates into which the flakes must be subdivided. Also, the greater the number of plates made in the treatment, the thinner they must be and, hence, the smaller the average diameter will tend to be.

We have seen that the value of the graphite for crucible manufacture depends largely on the diameter of the flakes composing the crucible stock. Therefore, if all of the interbedded impurities are to be removed, the resulting flakes will be so thin and of such small diameter that the value of the graphite for crucible making will be lessened.

Theoretically it would be possible so to separate the flakes into individual plates by buhr-mill grinding as to make a finished product in such a fine state of division that it will be valueless. Hence, the degree to which the finishing can be accomplished in a commercial plant will depend on the number and the characteristics of the interbedded impurities in the crude flakes. These factors will also determine the amount of grinding that is permissible in the buhr mill to permit removal of impurities, and will determine the grade of finished product that will be produced by a definite amount of such grinding. The proportion of dust produced will also depend on the same factors.

In other words, the interbedded material is the important factor in determining the economic possibilities of the finishing process. If the flake graphite contained in a certain crude concentrate is of such a nature that, in order to produce 90 per cent carbon in the finished product, an undue proportion of the carbon coarser than 100 mesh is destroyed, obviously the product will not be satisfactory.

There is no doubt that the buhr mill is the best commercial apparatus that has been developed to date for separating the flakes of graphite into their individual laminations and at the same time re-



ducing the freed impurities to a size permitting their removal by screening or bolting.

Several important features must be considered in grinding graphite concentrates in a buhr mill. The first point to be emphasized is that in order to produce a good crucible stock it is necessary to do more than remove the gangue present in the crude concentrates; it is also imperative that impurities interfoliated between the graphite laminations be removed. Clearly, in order to remove such impurities, some sort of grinding or rubbing process must be used, and, preferably, this should follow the removal of superficial or incidental impurities, so that these can not act as abrasive on the graphite flakes during the grinding.

From a survey of the grinding or polishing apparatus available, the conclusion is reached that the buhr mill is peculiarly adapted for this work, and that it is the only machine that can be depended on to do the required work in a satisfactory manner.

The results of the experiments indicate that the efficiency of the buhr mill, and hence of the whole finishing process, will depend on the character of the material that is being ground, and this in turn will depend both on the character of the flakes as they occur in the ore and on their treatment previous to grinding in the buhr mill. No hard and fast rules can be prescribed for the treatment of any particular concentrate, but in order to determine the most efficient treatment, it will be necessary to ascertain by experiment the best system to be followed.

The discussion indicates that satisfactory results depend on so many and such varied factors that undoubtedly there are certain concentrates produced from particular ore that can not be made into a crucible stock at a profit. Hence, it is true that although the equipment, the process, and the skill used in operation are all important for successful results, there is the added necessity that the flake itself meet certain requirements.

### IMPURITIES IN GRAPHITE.

In order to learn more of the chemical nature of the impurities that were encountered in the crude concentrates, and in the finished graphitic material, chemical analyses were made of ash samples from some of the representative graphite samples. The results are presented in the table following.

*Results of analyses of samples of graphite ash.*

Constituent.	Ash from graphite sample 2.	Ash from graphite sample 4.	Ash from sample 2 after special treatment. <sup>a</sup>	Ash from sample 2 after special treatment. <sup>b</sup>
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
SiO <sub>2</sub> .....	65.80	78.71	45.46	51.80
Al <sub>2</sub> O <sub>3</sub> .....	20.53	11.45	36.90	33.75
Fe <sub>2</sub> O <sub>3</sub> .....	10.77	8.65	14.00	11.85
CaO.....	.80	.73	.75	.67
Na <sub>2</sub> O and K <sub>2</sub> O.....	2.46	.71	2.68	2.54
Total.....	100.36	100.25	99.79	99.91

<sup>a</sup> Treated in electrostatic separator and in buhr mill.<sup>b</sup> Treated in buhr mill, given flotation treatment, and again put through buhr mill.

These results indicate that the finishing process tends to remove the silica but causes the alumina to be concentrated. This is, of course, due to the removal of the silica in a greater proportion than the mica, which contains the greater part of the aluminum oxide. This result is to be expected for the reason that the silica can be more easily ground because of its more massive form and brittle character.

It should be mentioned that mica, because of its flat shape and its toughness, is probably the most difficult impurity to remove in graphite refining.

**RESULTS FROM COMBINATION OF VARIOUS TREATMENTS.**

After considering the various apparatus that might be used in the production of crucible stock from crude concentrates, and a few of the factors that might be expected to affect the results, experiments were made to determine the results that could be obtained by logical combinations of the various schemes and apparatus.

The results already obtained indicated that two general schemes could be developed for a complete finishing process to produce crucible stock from Alabama concentrates. One of these schemes may be termed a wet process and the other a dry process.

The dry process is rather simple and involves only a treatment of the crude concentrate in the electrostatic separator, followed by grinding in the buhr mill and screening. The coarse sizes from the screen constitute the finished graphite suitable for crucible manufacture and these were found to carry 90 per cent carbon and flakes of the correct size for crucible manufacture.

The wet process is also simple, though involving one more step than the dry process. Wet concentrates are used in each step of this process, except that in the last step, grinding in the buhr mill, dry concentrates are used. Thus, the wet process should offer some advantages over the dry process. The wet concentrates are ground in the pebble mill for a period of time previously determined to be correct. The ground material is then floated in a standard flotation machine. The concentrates produced are dried and reground in the

buhr mill, after which they are screened. The screen used is 100 mesh and the oversize is the finished crucible stock.

A test of the dry process was made with sample 2, which assayed 51.23 per cent carbon. The material was twice treated in the electrostatic separator, a small sample was cut from the concentrates for analysis, and the remainder ground in the buhr mill. The product from this grinding was then screened and assayed. The whole product from the electrostatic treatment assayed 74.23 per cent carbon; the +100-mesh material contained 75.8 per cent and the -100-mesh substance, 59.8 per cent. The whole product from the grinding in the buhr mill assayed 78 per cent carbon; the -100-mesh material showed 45 per cent carbon, and the +100-mesh product, or crucible stock, 90.5 per cent. The total recovery of graphite in the finished crucible stock was 83 per cent of the graphite in the original sample.

The product obtained by grinding the concentrate from the electrostatic separator in the buhr mill was submitted to screen analysis and assay with the following results:

*Results of use of electrostatic separator and of buhr mill.*

Mesh.	Percentage of each size, by weight.	Carbon assay, (per cent).	Units of carbon, each size. <sup>a</sup>	Percentage of total carbon. <sup>b</sup>
+14.....	Tr.	.....	.....	.....
+20.....	Tr.	.....	.....	.....
28.....	0.9	92.8	83.00	1.06
35.....	5.1	91.8	468.00	6.00
48.....	15.2	91.4	1,388.00	17.80
65.....	23.0	90.1	2,054.00	26.30
100.....	26.3	86.2	2,265.00	29.00
150.....	13.8	85.6	1,181.00	15.10
200.....	4.3	39.8	171.00	2.20
-200.....	10.8	17.4	188.00	2.41
Total..	99.4	78.4	7,798.00	99.87

<sup>a</sup> Product of column 2 times column 3.

<sup>b</sup> Results obtained by dividing figures in column 4 by sum total of column 4.

The following tabulation shows calculated results based on the foregoing data and also the results of actual analyses:

*Selected data showing results of use of electrostatic separator and of buhr mill.*

Material.	Percentage of final product.	Carbon assay (per cent).	Units of carbon. <sup>a</sup>	Percentage of total carbon. <sup>b</sup>
Calculated results:				
Crucible stock (material over 100 mesh).....	70.5	88.8	6,258.0	80.2
Dust (material less than 100 mesh).....	28.9	53.3	1,540.0	19.8
Total.....	99.4	78.4	7,798.0	100.0
Results of analyses:				
Crucible stock.....	70.5	90.5	6,380.2	83.1
Dust.....	28.9	45.0	1,300.5	16.9
Total.....	99.4	77.2	7,680.7	100.0

<sup>a</sup> Column 2 times column 3.

<sup>b</sup> Results obtained by dividing figures in column 4 by sum total of column 4.

These results were exceptionally satisfactory. The grade of the coarse material was higher than the 90 per cent product required by the crucible makers, whereas the loss of less than 20 per cent of the total graphite in the original sample was lower than the loss in the finishing mills that are producing graphite of crucible grade from the Alabama concentrates.

As already mentioned, the whole success of a finishing process depends on the removal of a large percentage of coarse hard impurities that tend to cause excessive destruction of valuable flake in the buhr mill grinding. Such removal is, of course, the function of the electrostatic treatment, and the results prove how satisfactory such treatment was.

A test of the wet method was next made with 400 grams of sample 2 concentrate. The material was ground with 600 c. c. of water for one hour and floated in the Callow pneumatic machine. The concentrate from this treatment, when filtered and dried, contained 83 per cent carbon. This concentrate was put through the buhr mill, when the +100-mesh material was found to contain 90.3 per cent carbon. The recovery of the graphite was 80.2 per cent of the carbon in the original sample. Tabulated results follow.

*Results of treating concentrates in pebble mill, flotation machine, and buhr mill.*

Mesh.	Percentage, by weight, of each size.	Carbon assay (per cent).	Units of carbon, each size. <sup>a</sup>	Percentage of total carbon. <sup>b</sup>
14.....	None.			
20.....				
28.....	1.2	88.6	106.00	1.29
35.....	5.9	94.0	554.00	6.70
48.....	16.0	90.8	1,455.00	17.61
65.....	24.8	93.0	2,305.00	27.94
100.....	26.6	89.0	2,365.00	28.65
150.....	12.9	83.4	1,076.00	13.01
200.....	13.5	63.6	223.00	2.70
-200.....	10.1	17.4	167.00	2.02
Total..	100.0	82.5	8,251.00	99.91

<sup>a</sup> Column 2 by column 3.

<sup>b</sup> Results obtained by dividing figures in column 4 by sum total of column 4.

The following tabulation shows results based on the foregoing data:

*Selected data showing results of use of pebble mill, flotation machine, and buhr mill.*

Material.	Percentage of final products.	Carbon assay (per cent).	Units of carbon, each size. <sup>a</sup>	Percentage of original carbon. <sup>b</sup>
Crucible stock (material over 100 mesh).....	73.5	91.0	6,690.0	81.5
Dust (material less than 100 mesh).....	26.5	57.5	1,525.0	18.5
Total.....	100.0	82.5	8,215.0	100.0

<sup>a</sup> Column 2 times column 3.

<sup>b</sup> Results obtained by dividing figures in column 4 by sum total of column 4.



As with the dry process, the success of this treatment depends on the removal of the coarse impurities before the final buhr-mill grinding, and such removal is accomplished by the grinding in the pebble mill followed by flotation.

Owing to the short time available for the graphite investigation it was impossible to test all the samples with the combination treatment, but there can be no doubt that the results of such treatment would check closely with the results already described.

### GENERAL SUMMARY AND DISCUSSION OF RESULTS.

As a result of the investigations made to determine the most satisfactory methods of refining crude graphite concentrates, several important facts were developed.

Probably the most important of these is that each concentrate to be treated is a problem in itself, hence no treatment applicable to all kinds of concentrate can be prescribed. Such a limitation is due to the differences between the characteristics of the crude graphite flakes and those of their associated impurities. The most important characteristics that will affect the type of treatment to be applied are the hardness, toughness, diameter, and thickness of the flakes and their contents of interfoliated impurities. The different physical properties affect the amount and intensity of the grinding necessary to remove the impurities without the destruction of too large a proportional amount of the large and more valuable flakes.

Another important factor in determining a satisfactory finishing process is the kind of impurities that must be removed. When these impurities exist free from the flakes of carbon they can be removed, to a great degree, with a small loss of coarse carbon. When these impurities are soft and of small size they can be removed after a slight and nondestructive grinding operation, which will in all probability cause a comparatively small loss of valuable flake.

On the other hand, coarse, hard impurities are much more difficult to handle in a satisfactory manner. This type of material offers the greatest problem to the operator of the finishing mill. The results obtained in the experiments indicate that the success of the finishing process will depend on the success obtained in the removal of this sort of impurities at the earlier stages of the finishing operations.

The investigations have also brought out the fact that the buhr mill is a necessary apparatus in all processes for graphite finishing. The buhr mill separates the flakes into their constituent laminations and prepares them for the removal of the interbedded or interfoliated impurities.

The aspirator and the pneumatic jig were found to have an application under certain circumstances. It was seen, however, that such apparatus can be considered only for preliminary treatment previous to the regular finishing process and should be used only for the removal of very coarse, hard, free impurities that are encountered from time to time.

The experiments proved fairly conclusively that the use of either the electrostatic separator or the pebble mill, followed by flotation, will be applicable to all types of concentrates produced in the Alabama fields. The material produced by either of these types of apparatus can be ground in the buhr mill with satisfactory results.

The different processes of finishing graphite may be classified under two general heads—wet processes and dry processes. The decision as to which of the two should be used is dependent on the characteristics of the concentrates themselves. It is important that neither be adopted except after the necessary experiments and on the advice of an engineer thoroughly conversant with the problems to be encountered in the treatment of the particular concentrate in mind and with the economic conditions that will exist in the plant.

It can be said that with rare exceptions a satisfactory crucible stock can be produced from all of the concentrates that are at present made from the Alabama ores, and that the recovery of +100-mesh flake at the majority of finishing plants now operating can probably be improved. It should be possible to keep the recovery of material coarser than 100 mesh at 70 per cent of the total carbon in the original concentrates, and in many instances this can be increased to 80 per cent or more. These figures are sufficiently high to make profitable many of the plants that are not now operated except at a loss.

In conclusion, it should be emphasized that the results obtained in finishing the crude graphite concentrates are dependent largely on the character of the crude flake itself as contained in the ores. Moreover, mills that are producing a concentrate containing large tough flake in their concentrating plants, can also produce a satisfactory crucible stock at a profit from a finishing plant which has been correctly designed and intelligently managed. The operator who is unfortunate in having to mine small, low-grade flake requiring a large amount of intensive grinding, and which is associated with a large quantity of coarse siliceous or micaceous impurities, is bound to have difficulty in producing the required grade of crucible stock at a profit.

As in all mining ventures, success or failure depends more on the character of the raw ore than on the treatment given the ore after it has been mined.

## PUBLICATIONS ON MINING AND MILLING METHODS.

A limited supply of the following publications of the Bureau of Mines has been printed and is available for free distribution until the edition is exhausted. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest. Requests for publications should be addressed to the Director, Bureau of Mines.

The Bureau of Mines issues a list showing all its publications available for free distribution as well as those obtainable only from the Superintendent of Documents, Government Printing Office, on payment of the price of printing. Interested persons should apply to the Director, Bureau of Mines, for a copy of the latest list.

### PUBLICATIONS AVAILABLE FOR FREE DISTRIBUTION.

BULLETIN 75. Rules and regulations for metal mines, by W. R. Ingalls and others. 1915. 296 pp., 1 fig.

BULLETIN 80. A primer on explosives for metal miners and quarrymen, by C. E. Munroe and Clarence Hall. 1915. 125 pp., 51 pls., 17 figs.

BULLETIN 110. Concentration experiments on the siliceous red hematites of the Birmingham district, Ala., by J. T. Singewald, jr. 1917. 91 pp.

BULLETIN 128. Refining and utilization of Georgia kaolins, by I. E. Sproat. 1916. 5 pp.

BULLETIN 132. Siliceous dust in relation to pulmonary disease among miners in the Joplin district, Missouri, by Edwin Higgins, A. J. Lanza, F. B. Laney, and G. S. Rice. 1917. 116 pp., 16 pls., 6 figs.

BULLETIN 137. The use of permissible explosives in the coal mines of Illinois, by J. R. Fleming and J. W. Koster. 1917. 106 pp., 8 pls., 17 figs.

BULLETIN 139. Control of hookworm infection at the deep gold mines of the Mother Lode, California, by J. G. Cumming and J. H. White. 1917. 52 pp., 1 pl., 5 figs.

BULLETIN 154. Mining and milling of lead and zinc ores in the Missouri-Kansas-Oklahoma district, by C. A. Wright and H. A. Bueher. 1918. 134 pp., 17 pls., 13 figs.

TECHNICAL PAPER 105. Pulmonary disease in the Joplin district, Missouri, and its relation to rock dust in the mines, by A. J. Lanza and Edwin Higgins. 1915. 48 pp., 5 pls., 4 figs.

TECHNICAL PAPER 112. The explosibility of acetylene, by G. A. Burrell and G. G. Oberfell. 1915. 15 pp.

TECHNICAL PAPER 132. Underground latrines for mines, by J. H. White. 1916. 23 pp., 2 pls., 7 figs.

TECHNICAL PAPER 134. Explosibility of gases from mine fires, by G. A. Burrell and G. G. Oberfell. 1916. 31 pp., 1 fig.

TECHNICAL PAPER 154. Suggestions for improved methods of mining coal on Indian lands in Oklahoma, by J. J. Rutledge and Daniel Harrington. 1918. 32 pp.

TECHNICAL PAPER 174. Suggestions for the safe operation of gasoline engines in mines, by R. H. Kudlich and Edwin Higgins. 1917. 19 pp., 3 figs.

TECHNICAL PAPER 182. Flotation of chalcopyrite in the chalcopyrite-pyrrhotite ores of southern Oregon, by W. H. Coghill. 1917. 13 pp.

MINERS' CIRCULAR 5. Electrical accidents in mines, their causes and prevention, by H. H. Clark, W. D. Roberts, L. C. Ilsley, and H. F. Randolph. 1911. 10 pp., 3 pls.

MINERS' CIRCULAR 10. Mine fires and how to fight them, by J. W. Paul. 1912. 14 pp.

MINERS' CIRCULAR 11. Accidents from mine cars and locomotives, by L. M. Jones. 1912. 16 pp.

MINERS' CIRCULAR 13. Safety in tunneling, by D. W. Brunton and J. A. Davis. 1913. 19 pp.

MINERS' CIRCULAR 17. Accidents from falls of rock and ore, by Edwin Higgins. 1914. 15 pp., 8 figs.

MINERS' CIRCULAR 18. Notes on miners' carbide lamps, by J. W. Paul. 1915. 11 pp.

MINERS' CIRCULAR 19. The prevention of accidents from explosives in metal mining, by Edwin Higgins. 1914. 16 pp., 11 figs.

#### PUBLICATIONS THAT MAY BE OBTAINED ONLY THROUGH THE SUPERINTENDENT OF DOCUMENTS.

BULLETIN 53. Mining and treatment of feldspar and kaolin in the Southern Appalachian region, by A. S. Watts. 1913. 170 pp., 16 pls., 12 figs. 35 cents.

BULLETIN 107. Prospecting and mining of copper ore at Santa Rita, N. Mex., by D. F. MacDonald and CharlesENZIAN. 1916. 122 pp., 10 pls., 20 figs. 25 cents.

BULLETIN 111. Molybdenum; its ores and their concentration, with a discussion of market prices, and uses, by F. W. Horton. 1916. 132 pp., 18 pls., 2 figs. 30 cents.

BULLETIN 121. The history and development of gold dredging in Montana, by Hennen Jennings, with a chapter on placer-mining methods and operating costs, by Charles Janin. 1916. 64 pp., 29 pls., 1 fig. 30 cents.

BULLETIN 127. Gold dredging in the United States, by Charles Janin. 1918. 224 pp., 63 pls., 23 figs. 50 cents.

BULLETIN 146. The technology of salt making, by W. C. Phalen. 1917. 144 pp., 24 pls., 10 figs. 25 cents.

BULLETIN 153. The mining industry in the Territory of Alaska during the calendar year 1916, by Sumner S. Smith. 1917. 91 pp., 1 pl. 10 cents.

BULLETIN 160. Rock quarrying for cement manufacture, by Oliver Bowles. 1918. 160 pp., 6 pls., 31 figs. 25 cents.



TECHNICAL PAPER 17. The effect of stemming on the efficiency of explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp., 11 figs. 5 cents.

TECHNICAL PAPER 24. Mine fires, a preliminary study, by G. S. Rice. 1912. 51 pp., 1 fig. 5 cents.

TECHNICAL PAPER 41. The mining and treatment of lead and zinc ores in the Joplin district, Mo., a preliminary report, by C. A. Wright. 1913. 43 pp., 5 figs. 5 cents.

TECHNICAL PAPER 67. Mine signboards, by Edwin Higgins and Edward Steidle. 1913. 15 pp., 1 pl., 4 figs. 5 cents.

TECHNICAL PAPER 95. Mining and milling of lead and zinc ores in the Wisconsin district, Wisconsin, by C. A. Wright. 1915. 39 pp., 2 pls., 5 figs. 10 cents.

TECHNICAL PAPER 222. Methods of administering leases of iron-ore deposits belonging to the State of Minnesota, by J. R. Finlay. 1919. 40 pp., 1 fig. 5 cents.



# INDEX.

A.	Page.		Page.
Alabama, graphite refining methods		Concentration tests, various treat-	
in -----	46	ments combined -----	70
graphite industry in -----	11, 42	with aspirator -----	51
Alling, H. L., work cited -----	9	with electrostatic separator -----	58
Air classification, description of -----	24	with pebble mill -----	54
Air classifier, figure showing -----	25	with pneumatic jig -----	53
sampling with, results -----	15	Crushing methods, in Alabama, de-	
<i>See also</i> Aspirator.		scription of -----	13
Analyses. <i>See</i> Concentrates; Graph-			
ite; Screen analysis.		E.	
Ash, in graphite sample, determina-		Electrostatic separator, description	
tion of -----	44	of -----	59-60
analysis of, results of -----	70	concentration tests with -----	58-63
Aspirator, concentrates from, screen		tests with, results of -----	62, 63
analyses and assays of -----	51	transformer readings -----	62
concentration tests with -----	51	figure showing -----	61
description of -----	50	view of -----	60
view of -----	50	wiring diagram for -----	62
<i>See also</i> Air classifier.		Electrostatic separator and buhr mill,	
		results with -----	71
B.			
Buhr mill, use in graphite finishing -----	66	F.	
views of -----	50, 53	Ferguson, H. G., work cited -----	8, 11
Buhr mill and electrostatic separ-		Flotation, graphitic ores, success of -----	65
ator, results with -----	71		
Bureau of Mines, analysis of graph-		G.	
ite, method used by -----	43	Graphite, analysis of, Bureau of	
electrostatic separator designed		Mines method for -----	43
by -----	61	ash in analysis of -----	70
		chemical analysis of -----	40
C.		impurities in -----	67, 69
Callow pneumatic cells, sampling		microphotographs showing -----	66
with, results of -----	19	literature on -----	8
Callow pneumatic flotation system,		No. 1 flake, specifications for -----	38
flow sheet of -----	21	occurrence, Alabama -----	11
Carbon, in graphite, determination		New York -----	28
of -----	45	Pennsylvania -----	30
Cirkel, Franz, work cited -----	8	Texas -----	33
Colmer washer, description of -----	16	physical properties of -----	49
section of, figure showing -----	16	screen analyses of -----	39
Concentrates, crude, refoating tests		specifications for -----	40
of, results of -----	66	shipments of, embargo on -----	9
from aspirator, screen analyses		structure of, microphotographs	
and assays of -----	51	showing -----	67
from pneumatic jig, screen anal-		Graphite Producers Association,	
yses and assays of -----	54	acknowledgment to -----	8
used in tests, screen analyses		Graphite products, recovery of, in	
and assays of -----	48	Alabama -----	26
Concentrating plant, erection of,		Graphitic carbon, determination of -----	45
cost of -----	27		
New York, flow sheet of -----	29	II.	
Concentration methods, description		Huff, electrostatic refining system,	
of, Alabama -----	13	flow sheet for -----	31
Pennsylvania -----	32		
New York -----	29	J.	
Texas -----	34	Jig. <i>See</i> Pneumatic jig.	

K.	Page.			Page.
Kish, recovery of graphite from-----	35	Pebble mill, wet grinding in, de- scription of-----	54	
L.		results of, curves showing-----	56, 57	
Log-washer process, description of--	17	results of, tables of-----	56, 58	
flow sheet of-----	18	Pebble mill, flotation machine, and buhr mill; results with-----	72	
<i>See also</i> Concentration methods.		Q.		
M.		Quartz, physical properties of-----	49	
Mica, physical properties of-----	49	R.		
Microphotographs, impurities in graphite ore, view of-----	66	Refining methods, description of, Alabama-----	23, 46-74	
structure of graphite ore, view of-----	67	New York-----	30	
Miller, B. L., work cited-----	8	Refining plant, flow sheet of-----	24	
Minerals separation system, flow sheet of-----	20	Reflotation tests, results of-----	65	
Mining methods, description of, Ala- bama-----	11	S.		
New York-----	28	Sample, preparation for analysis of--	44	
Pennsylvania-----	32	Sampling, sacked graphite, tests of, results of-----	37	
Texas-----	33	Sampling device, sections of, figure showing-----	38	
Moisture, determination of-----	44	Sampling methods, description of--	36	
Mudd, H. S., acknowledgement to--	8	diagram of, figure showing-----	36	
Munro washer, new, description of--	17	Screen analysis, specifications for--	40	
section of, figure showing--	17	<i>See also</i> Aspirator; Graphite samples; Pneumatic jig; Pebble mill.		
original, description of-----	16	Separator. <i>See</i> Electrostatic sepa- rator.		
section of, figure showing--	16	Simplex flotation system, description of-----	19	
<i>See also</i> Colmer washer; Log washer.		flow sheet of-----	22	
N.		T.		
New York, graphite industry in-----	27, 42	Texas, graphite industry in-----	33, 43	
O.		Transformer readings, in electro static-separator tests, re- sults of-----	62	
Oil-froth flotation systems, descrip- tion of-----	18	V.		
Operating costs, United States, 1918- 41		Volatile matter, determination of--	44	
Oscillators, sampling with, results of-----	32	W.		
P.		War Industries Board, control of shipments by-----	10, 41	
Paige, Sidney, work cited-----	33	War Trade Board, ruling on ship- ments of-----	10	
Pennsylvania, graphite industry in- 30, 42		Water "skin flotation" system, de- scription of-----	14	
Pneumatic jig, concentrates from, screen analyses and as- says of-----	54.	flow sheet of-----	14	
concentration tests with, de- scription of-----	53			
description of-----	53			
view of-----	53			



DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

JUN 10 1920

*in full in*

# MINING AND PREPARING DOMESTIC GRAPHITE FOR CRUCIBLE USE

BY

GEORGE D. DUB and FREDERICK G. MOSES

WITH A

CHAPTER ON METHODS OF ANALYSIS  
USED BY THE BUREAU OF MINES

BY

G. B. TAYLOR and W. A. SELVIG



WASHINGTON  
GOVERNMENT PRINTING OFFICE

1920



















UNIVERSITY OF ILLINOIS-URBANA



3 0112 001838280